SEDGMAN

FINAL REPORT

NI43-101 Technical Report

Corani Project Detailed Engineering Phase 1 (FEED)

BEAR CREEK
Mining Corporation

Bear Creek Mining Corporation

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1 Executive Summary

This report is developed for Bear Creek Mining Corporation by Sedgman. The document is elaborated with the information generated during the FEED, Detail Engineering Phase 1 carried out by Graña Montero Engineering (GMI). The technical report presented is an international recognized bankable document named as National Instrument 43-101 Project Report (NI 43-101) that summarizes the FEED phase update.

This Report is based on the outcomes of the Feasibility Study (FS) carried out by M3 Engineering & Technology Corp. (M3), complemented by a detail engineering phase 1 by GMI.

The Project has been modified in order to optimise the facilities or reduce the costs. The main changes compared with the 2015 report are presented as follow:

- Optimization of the Mining Plan.
- Geo referenced topography and with greater precision.
- Update of Geometallurgical model for predicting recovery within the block model.
- Change in the design of the Primary Crusher to Gyratory type.
- Update of Process Design Criteria and Plant Water Balance.
- Reducing the Stockpile capacity.
- Studies of Soil Mechanics.
- Review of bridge and tunnel designs in the Main Project Access.
- Evaluation of the integral Project Layout.
- Update of Capex and Opex.
- Update of local construction costs.
- Development of the Project Execution Schedule.
- Development of the WBS for the Project.
- Review and update of Hydrology and Hydrogeology of the Project.
- Development of designs which allow to confirm of quantities for items with high cost incidence.
- Evaluation and development of Project accesses.
- Development of accesses for mine vehicles (Haul Roads).

1.1 PROPERTY AND LOCATION

The land status of the Corani Project is a series of twelve (12) mineral claims or concessions. Mineral concessions in Peru are filed with the Instituto Nacional de Concesiones y Catastro Minera (INACC) which is part of the Ministerio de Energía y Minas in Peru (MINEM). Claims can vary in size from 100 to 1,000 hectares (ha). They are rectangular geometries parallel to the UTM grid system employed in the district. The Corani Project is located in the district of Corani, province of Carabaya department of Puno, in Peru, and covers an aggregate extent of 5,180.1213 hectares. The concessions are fully controlled by BCM and are free of any mortgage, lien, charge, royalty, or encumbrance.

The twelve (12) mineral concessions comprising the Project are subject to compliance with payment of annual license fees in the amount of US$3.00 per hectare (“License Fees”). In addition, they are subject to an annual maintenance requirement with either of the following alternative obligations: minimum required levels of annual production of at least US$100 per ha in gross sales (“Minimum Production”); or payment of an additional amount referred as Penalty of US$6.00 per ha for the 7th through 11th year following the granting of the concession, and of

US$20.00 per ha thereafter; or exploration expenditures of 10 times the Penalty. Compliance with one of these three maintenance obligations, together with timely payment of License Fees, is required by them in good standing. Failure to comply with License Fee payments or Penalty payments for two consecutive years causes the forfeiture of the mineral concessions.

In the year 2018, the twelve (12) mineral concessions comprising the Project shall be subject to the obligations of Minimum Production, Penalties and exploration expenditures in accordance with the maintenance regime in force as of October 2008 whereby:

- The minimum production will be equivalent to one Tax Unit per year (approximately U.S. $1,333.00) per hectare granted for metallic minerals, and 10% of one Tax Unit per year per hectare granted for non-metallic minerals ("Minimum Production").
- Failure to attain Minimum Production will trigger the obligation to pay a penalty equivalent to 10% of the Minimum Production per year per hectare, until the year in which the Minimum Production is attained.
- Year 2028 shall be the maximum deadline for the mineral concessions comprising the Corani Project to attain Minimum Production. Failure to do so will result in the forfeiture of these mineral concessions.

Control and current status were verified in August 2017 through an electronic database search of the Geologic Mining and Metallurgical Institute (INGEMMET). All concessions are in good standing.

1.2 ACCESSIBILITY, CLIMATE

The Project site is located in the eastern Andes mountain range, between 4,600 and 5,200 m above sea level (masl). The area is characterized by mountainous terrain dominated by volcanic rock above which sits glacial gravel. The lithologic and climatic conditions have given rise to a series of cirques or bowl-shaped, steep-walled basins. Apart from the vegetation associated with the wetlands mentioned below, areas of "puna" or alpine tussock grassland occupy the valleys and moderate to steep slopes. The areas above 4,700 masl mostly consist of steep mountainous slopes where erosion and climatic conditions largely prevent the development of soils or vegetation. These areas are scarcely vegetated with species specially evolved to withstand the harsh conditions. The naturally occurring acidic soils related to oxidation of sulfide bearing materials and the resulting ARD from exposed mineralized zones within the Project area have also prevented the development of vegetation where these conditions occur.

Existing access to the Project site is primarily by road from the town of Macusani (located on the paved dual lane Interoceanic Highway), which is more readily accessible from the town of Juliaca, also serviced by commercial airlines from Lima. This route typically takes 4.5 to 5 hours. There are other access routes to the site from Cusco, taking approximately 6 hours by vehicle on increasingly primitive roads approaching the site. The City of Cusco is also serviced by commercial airlines from Lima.

The nearest town of significant size and with significant infrastructure is Macusani, which is the capital of the Province of Carabaya in the Region of Puno. Macusani is approximately 30 km east of the Project in a direct line. The access road from Huiquisa Bridge to the Permanent Camp will be improved. The length of the proposed Mine Access Road connecting the process Plant to Macusani is anticipated to be approximately 64 km. The Project site is located in the district of Corani, also in the Province of Carabaya. The closely orientated campesino communities of Chacaconiza and Quelcaya, which have a joint population of approximately 200 families (80 and 120 families, respectively), will be directly impacted by the mine development, in terms of landholding, rights to water, employment, etc. The community consultation undertaken with the Chacaconiza and Quelcaya communities to date has included a proposal for mining employment, generating widespread acceptance, mainly among younger community members, the teachers at local educational facilities, and community leaders. The current labor force is generally unskilled, mainly working on highway remediation and maintenance. A technical training program is planned to develop the skills of community members to fulfil employment requirements of the Project. The Project's requirement for labor will exceed the labor resources available in the Chacaconiza and Quelcaya communities. A ranking system will be developed with regard to geographical location of employment applicants, together with categorization and quantification of the labor force required.

1.3 HISTORY

Prior to the early 1950s, mineral exploration in the Corani district consisted of shallow prospect pits and adits in the northern portion of the current Project area. These prospects are of unknown age and may date back to colonial Spanish time. Antimony prospects south and east of the property reportedly were active in the early 1900s, when there was limited antimony production (C.R. Petersen, 1967).

The first modern evaluation of silver-lead mineralization began with the location of mineral concessions in 1951, and in 1956 Compañía Minera Korani was formed to develop the silver-lead mineralization previously prospected. The mines were developed and operated from 1956 to at least 1967. Total historical production is uncertain, but is estimated at 100,000 tonnes of silver-lead-zinc ore. In early 1967, estimated mine production was reported at about 3,400 short tons per month, with grades of 7.0-9.0% lead, 2.3% zinc, and 8.0 to 11.0 oz/ton silver (C.R. Petersen, 1967).

The next exploration activity was by a private Peruvian company, Minsur. That exploration was reported to include 40 shallow drill holes in various locations, including a number of close proximity holes in the gold zone (located south of the current resource area). Although Minsur is an active mining company in Peru, attempts by BCM to secure copies of Minsur's exploration data have been unsuccessful. None of Minsur's exploration information is available or verifiable, although, reportedly, gold mineralization was encountered in some of Minsur's drilling.

In late 2003 and early 2004, Rio Tinto Mining and Exploration began a surface exploration program for porphyry copper mineralization. That initial work by Rio Tinto defined anomalous silver and lead mineralization to the south of the Korani mines and also defined a zone of anomalous gold mineralization in rock and soils. The concession ownership by Compañía Minera Korani apparently lapsed during the 1970s. The ownership of Minsur also lapsed prior to Rio Tinto's exploration activities after 2000. Rio Tinto re-established some of the older concessions in their name beginning in 2003. BCM has added two concessions early in 2005 to create the current land position described in Section 4.

Six previous resource estimates and two previous mineral reserve estimates have been completed for the Project and are published in previous technical reports beginning in 2006. Since 2006, the Measured and Indicated resource has grown from approximately 40 Moz of silver to over 300 Moz of silver.

1.4 GEOLOGICAL SETTING AND MINERALIZATION

The Project area is underlain by Tertiary volcanic rocks of the Quenemari Formation, specifically a thick series of crystal-lithic tuffs and andesite flows, which overlie variably deformed Lower Paleozoic to Mesozoic metasediments of the Ambo and Tarma Groups. The primary host of mineralization is the Chacacuzina Member of the Quenemari Formation. The Chacacuzina is the youngest member of the Quenemari, and is comprised of a sequence of crystal-lithic and crystal-vitric-lithic tuffs. The tuffs are widely hydrothermally altered and pervasively argillized to low-temperature clays, and are variably faulted, fractured, and brecciated.

Mineralization at the Corani Project occurs in three distinct and separate zones: Corani Main, Corani Minas, and Corani Este, each differing slightly in character with regard to both alteration and mineral assemblages. In general, mineralization in outcrops throughout the Corani Project is associated with iron and manganese oxides, barite, and silica. Silicification is both pervasive and structurally controlled along veins. In drill core, the mineralization occurs in typical low to intermediate sulfidation Ag-Pb-Zn mineral assemblages. The most abundant silver-bearing mineral is fine-grained argentian tetrahedrite or freibergite.

Structurally, the Corani deposit is situated within a stacked sequence of listric normal faults striking dominantly north to north-northwest with moderate to shallow (50° to <10°) westerly dips. The hanging walls of the listric faults are extensively fractured and brecciated, providing the structural preparation for subsequent or syngenetic mineralization. The stacked listric faults are more prominent in the Corani Minas and Corani Main areas. The Corani Este area contains a single known listric fault with an extensively fractured and brecciated hanging wall. The contact with the underlying Paleozoic sediments corresponds locally to listric faults dipping shallowly to the west.

1.5 DEPOSIT TYPES

The Corani deposit is best described as a low- to intermediate-sulfidation epithermal deposit with silver, lead, and zinc mineralization hosted in stock works, veins, and breccias. Mineralization is principally located in a set of lístric faults dipping west, with dilational segments related to subvertical structures and breccias in the hanging wall, and veinlets forming stockworks in the footwall. Structural control of the mineralization is a product of extensional tectonics that developed the series of north- to northwest-trending fractures and faults, and whose movements provided the structural preparation for the influx of mineralizing hydrothermal fluids.

Mineralization at Corani is likely both laterally and vertically distal to an intrusive fluid source. Mineral textures grade from coarse crystalline quartz-pyrite-chalcopyrite in the southern portion of the Project area, to finer grained, pyrite-dominated sulfide minerals in the north, suggesting a south-to-north hydrothermal fluid flow. This spatial zonation suggests a rapidly cooled ore fluid typical of a distal setting surrounding a buried intrusion. The multiphase nature of the mineralization and zonation at Corani may be related to multiple fluid exsolution events from an evolving porphyry type system that possibly underlies the southern part of the area. Alternatively, the mineralizing solutions may be related to shallow, subvolcanic dome emplacement.

1.6 EXPLORATION

BCM began exploring the Corani Project in early 2005. In addition to drilling, exploration activities carried out by BCM include detailed geologic mapping, trenching, and geophysical surveying.

BCM has conducted general geologic surface mapping over the entire Project area. The total mapped surface is about 4.5 km wide (east-west) and 7.5 km long (north-south). In 2015, detailed surface mapping, including lithology, alteration, and structures, was performed at a scale of 1:2500 in the area of the proposed pits.

BCM has completed 25 trenches within the Project resource area (Corani Main, Minas, and Este) to verify the continuity of the structures covered by Quaternary sediments. Spacing between the trenches is roughly 50 to 100 meters. Channel samples from these trenches have produced an associated 1,295 assay intervals for a total of 2,924 meters of trench data.

VDG del Perú S.A.C. (VDG) conducted a ground geophysical campaign at the Corani Project on behalf of BCM in the fall of 2005. A total of 44.20 line-km of induced polarization (IP) data was collected, along with 50.95 line-km of magnetic survey. The geophysical surveys were aimed at assisting in geological mapping, including lithologies and key structures and at mapping mineralization and alteration associated with a low sulfidation gold-silver system. The objective of the IP/Res survey was to map the electrical response by means of high-resolution IP traverses across the favorable north-south corridor identified based on the results of both trench and drilling exploration. The field results of both methods were of good quality and were meaningful. The final chargeability and resistivity depth sections mapped systematically clear contrasts from line to line between the sub-surface and a nominal depth of 283 meters below surface. The chargeability outlined five (5) IP anomalies, two of which correspond to the Corani Main and Corani Este areas, respectively. Those anomalies accurately mapped the known mineralization and extended the size of both mineralized zones.

1.7 DRILLING

Since 2005, BCM has completed a total of 556 drill holes at the Corani Project for a total of approximately 100,494.57 m. Drilling was completely by the Peruvian contractor, Bradly MDH primarily using LD250, JKS35, and LJ44 drill rigs. All of the drilling to date has been completed using diamond core drilling methods to produce either HQ (6.35 cm dia.) or NQ (4.76 cm dia.) core. Diamond drill hole data contained in the Project database to date includes all 556 drill holes with an associated 38,111 sample intervals over a total of 92,742 m of drilling. The Project database contains 36,996 assay values each for silver, lead, zinc, and copper.

1.8 SAMPLE PREPARATION, ANALYSES AND SECURITY

BCM employs standard, basic procedures for both drill core and trench sample collection and analysis. Formal chain of custody procedures are maintained during all segments of sample transport. Samples prepared for transport to the laboratory are bagged and labelled in a manner which prevents tampering, and remain in BCM control until released to private transport carrier in Cusco or Juliaca. Upon receipt by the laboratory, samples are tracked by a blind sample number assigned and recorded by BCM. The samples are prepped according to ALS-Chemex preparation code PREP-31, and silver, lead, zinc, and copper assays are carried out by three-acid digestion followed by atomic absorption spectrophotometry (AA) analysis. Multi-element inductively coupled plasma (ICP) analysis is conducted on select sample intervals to assist with mineralization classifications and to guide the interpretation of the metallurgical process response.

BCM maintains an internal Quality Assurance/Quality Control (QA/QC) program which includes both standard and check (lab) sampling. GRE conducted a critical review of BCM's QA/QC program; toward that end, BCM provided GRE with QA/QC data in multiple Excel spreadsheet files. GRE compiled the data into a single, comprehensive QA/QC data worksheet for analysis and evaluation. Based on the results of GRE's review, in conjunction with observations and conversation with BCM personnel during the QP site visit, BCM's routine sample preparation, analytical procedures, and security measures are, in general, considered reasonable and adequate to ensure the validity and integrity of the data derived from BCM's sampling programs. GRE recommends that BCM expand the existing QA/QC program to include at least standards, blanks, and duplicates, and that QA/QC analysis be conducted on an on-going basis, including consistent acceptance/rejection tests. Each round of QA/QC analysis should be documented, and reports should include a discussion of the results and any corrective actions taken.

1.9 DATA VERIFICATION

Data verification efforts included an on-site inspection of the Corani Project and core storage facility, check sampling, and manual and mechanical auditing of the Project database.

During the on-site inspection in August 2017, GRE's (QP) representative conducted general geologic field reconnaissance, including inspection of bedrock exposures and other surficial geologic features, ground-truthing of reported drill collar and trench sample locations, and superficial examination of historic mine workings. One full day of the site visit was spent at the core storage facility in Juliaca, where select intervals of whole and half core were visually inspected and samples were selected to submit for check assay. Field observations during the site visit generally confirm previous reports on the geology of the Project area. Bedrock lithologies, alteration types, and significant structural features are all consistent with descriptions provided in existing Project reports, and the author did not see any evidence in the field that might significantly alter or refute the current interpretation of the local geologic setting.

Specific core intervals from 35 separate drill holes were selected for visual inspection and potential check sampling based on a preliminary review of the drill hole logs and associated assay values. The core intervals were selected prior to the site visit, and the core was laid out by BCM staff and ready for inspection upon arrival. With few exceptions, the core samples accurately reflect the lithologies recorded on the logs. A total of 17 samples were selected for check assay. The samples were selected from low, moderate, and high-grade intervals based on original assay results. In all cases, the degree of visible alteration and evidence of mineralization observed was generally consistent with the grade range indicated by the original assay value. Laboratory analysis was completed by ALS Peru using the same sample preparation and analytical procedures as were used for the original samples. Standard t-Test statistical analysis was completed to look for any significant difference between the original and check assay population means. The results of the t-Test showed no statistically significant difference between the means of the two trials (original versus check assay).

GRE completed a QAQC audit of the digital Project database by comparing a random selection of original assay certificates to the assay information contained in the Corani Project database. Results of the QAQC audit indicate a minor and acceptable error rate. GRE also completed a mechanical audit of the Project database in order to evaluate the integrity of data from a data entry perspective. The mechanical audit identified a small number of data entry errors, including gaps, overlaps, and missing sample intervals. All data entry errors were easily rectified, and are considered insignificant with regard to potential impact to the mineral resource and mineral reserve estimates. The database audit work completed to date indicates that occasional inconsistencies and/or erroneous entries are likely inherent or inevitable in the data entry process.

GRE recommends that BCM establish a routine, internal mechanical audit procedure to check for overlaps, gaps, total drill hole length inconsistencies, non-numeric assay values, and negative numbers. The internal mechanical audit should be carried out after any significant update to the database, and the results of each audit, including any corrective actions taken, should be documented and stored for future use in database validation.

1.10 MINERAL PROCESSING AND METALLURGICAL TESTING

The Corani deposit is a silver-lead-zinc deposit with relatively complex mineralogy. It should be noted that no additional metallurgical work has been completed since issuance of the 2015 Optimized Feasibility Study. The geo-metallurgy recovery equations developed for the 2015 study were utilized for the current mine planning and scheduling. Upon review of the metallurgical testing data, it is clear that performance of Corani mineralization to conventional flotation and cyanidation processing was widely variable. The geological classifications provided some delineation of metallurgical response: Samples representing the CSC ore type consistently responded well to a conventional sequential lead / silver – zinc flotation. Conversely, FeOx and MnO responded very poorly to flotation, but generally responded better to cyanide leaching for silver.

The geological classification FBS which represents a large amount of the estimated resource tonnage, had a broad range of metallurgical response. The variable response was shown to be related to the fine texture of the mineralization and presence of non-sulphide lead mineral forms. However, the geological classifications alone were not able to delineate the texture or quantity of non-sulphide lead minerals.

To better predict the metallurgical response, a geo-metallurgy approach was investigated to link metallurgical response to block modelling parameters. The statistical analysis indicated several key parameters could be used to generate metallurgical response. For the purpose of the analysis, the metallurgical process was restricted to only sequential flotation of silver bearing lead concentrate followed by the flotation of a zinc concentrate, also containing some silver.

With metallurgical response linked to block modelling parameters, the mine plan could be optimized to maximize the revenue for the Project. The Table 1-1 displays the estimated metal recoveries by mine schedule.

Table 1-1: Recovery Predictions for Mine Schedule

Production Year	Tonnes t(dry) x 1000	Feed Grade (% or g/t)		Grade (g/t or %)		Recovery (%)	
		Ag	Pb	Ag	Pb	Ag	Pb
Year 1	5,328.75	95	1.14	4,560	50.0	65.6	59.8
Year 2	7,875	77	1.06	3,569	49.9	75.4	66.3
Year 3	7,875	93	1.24	4,370	53.8	67.7	62.4
year 4 to 5	15,750	62	1.10	3,086	53.8	68.5	67.3
Year 6 to 10	39,375	51	0.90	3,858	50.0	60.4	51.7
Year 11 to 18	62,868.9	35	0.76	2,354	50.2	64.2	64.5
LOM	139,072.6	50	0.90	3,010	51.0	64.3	61.1

1.11 MINERAL RESOURCE ESTIMATES

GRE estimated the Mineral Resources for the Corani Project during the first quarter of 2015. No new drilling, geology, or metallurgical test work has been performed since then. That work and the resulting Mineral Resources were documented and published in the May 30, 2015 Technical Report. The 2015 mineral resource block model was used for estimation of the Mineral Resources and Mineral Reserves of the Corani Project in the current Corani Project Detailed Engineering Phase 1 (FEED) Technical Report.

The resource model has three main lithologies: basement sediment with minor quantities of mineralization, the mineralized (pre-mineral) tuff, and a mostly unmineralized post-mineral tuff which is assumed to be barren. Mineralization has been defined by 7 mineralization types, which were later grouped into oxidized, transition, and sulphide groups. The Mineral Resources for the Corani Project are shown in Table 1-2. The Mineral Resources were generated within the $30.00/troy ounce silver, $1.425/lb lead, and $1.50/lb zinc price Whittle pit shell and the calculated $11/tonne NSR cut-off.

Table 1-3 shows the potentially leachable Mineral Resource contained within the Whittle pit shell at a 15 g/t cut-off that is available in addition to the Mineral Resource shown in Table 1-2.

Table 1-2: Total Mineral Resources (Includes Both Resources and Reserves)

Category	Ktonnes	Silver gpt	Lead %	Zinc %	Silver Million oz	Lead Million lb	Zinc Million lb
Measured	29,209	56.2	0.912	0.582	52.8	587	375
Indicated	181,902	40.7	0.741	0.495	238	2971.3	1983.5
Measured + Indicated	211,111	42.8	0.765	0.507	291	3,558	2,359
Inferred	31,231	40.6	0.742	0.512	40.8	510.6	352.4

Note: Cut-off Value : $11.00/tonne covers process and general and administrative costs.

Table 1-3: Total Mineral Resource of Potentially Leachable Material

Category	Ktonnes	Silver gpt	Silver Million oz
Measured	5,006	38.0	6.12
Indicated	19,690	23.1	14.61
Measured & Indicated	24,697	26.1	20.72
Inferred	8,722	25.1	7.03

1.12 MINERAL RESERVE ESTIMATES

The Mineral Reserve Estimate is based on the 2015 GRE resource block model, using updated Whittle optimization parameters, new pit designs, and new phase designs.

The Project Mineral Reserves consider only measured and indicated resource categories, which have been converted to proven and probable reserves categories, respectively. Mineral Reserves are defined as being the material to be fed to the process plant in the mine plan already described, and are demonstrated to be economically viable in the Detailed Design Phase 1 (FEED) economic model.

Table 1-4: Corani Project Mineral Reserves

Classification	Tonnes Mt (dry)	Grade			NSR $/t	Contained Metal		
		Silver g/t	Lead %	Zinc %		Silver Moz	Lead Mlb	Zinc Mlb
Proven	20.8	65.8	1.03	0.71	37.17	44	472	323
Probable	118.3	47.5	0.87	0.57	28.55	181	2,274	1,486
Total Proven + Probable	139.1	50.3	0.90	0.59	29.84	225	2,746	1,809

Notes:

1) The Mineral Reserves have been estimated using the definitions of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).
2) The Mineral Reserves have been estimated using the following metal prices: $20.00/oz Ag, $1.00/lb Zn, $0.95/lb Pb using a revenue factor 1.00 pit shell as a basis for the pit design.
3) Only pre-mineral tuff type of material has been considered as reserves.
4) NSR Cut-off grades used are equal or higher than: $11.11/t for the East Pit, and $11.26/t for Minas and Main pits.
5) The effective date for these Mineral Reserves is 1 May 2017.
6) Totals / Averages may not add up due to rounding of individual tonnes and grades.
7) The tonnes and grades shown above are considered a Mineral Reserve because they have been demonstrated to be economically viable through the FEED study financial model using the following metal prices: $18.00/oz Ag, $1.10/lb Zn, $0.95/lb Pb.

1.13 MINING METHODS

The Corani Project will be mined using conventional open pit mining methods, with either an owner mining or a contractor mining scenario. The base case assumes contractor mining. The rock will be broken by drilling 0.156-m diameter blast holes and blasting with ANFO and emulsion. Broken rock will then be loaded into 130-136 tonne trucks using a 13 m^3 front end loader or one of two 15 m^3 hydraulic shovels. Support equipment includes two D10 bulldozers, a road grader, water trucks, rubber tire dozer, compactor, excavator, fuel and lube trucks, and other miscellaneous equipment.

During a 10-month pre-production stripping, pioneering, haul road construction phase prior to plant construction, 4.7 million tonnes of waste rock will be mined to generate construction material. Another 7.6 million tonnes will be mined immediately prior to production. The mine is designed to generate 7.875 million tonnes of ore per year with strip ratios of 3:1 to 4:1 during the first two years, falling to below 2:1 the third year, and then below 1:1 the final 3 years of the mine life.

1.14 RECOVERY METHODS

Corani process plant is designed for a treatment of 7,875,000 tonnes per year of a lead-silver-zinc ore, considering an availability of 92% and operating 350 days per year. It considers typical mining industry areas such as crushing, grinding, flotation, thickening and filtration for tailings and final concentrates (Pb/Ag and Zn/Ag).

The plant design considers the following process areas:

- Primary Crushing and Coarse Ore Stockpile
- SAG – Ball Mill
- Selective Lead and Zinc Flotation
- Concentrate Thickening and Filtering (Pb, Zn)
- Tailings Thickening and Filtering Plant
- Dry Tailings Disposal Plant
- Reagents and Utilities Areas.

Figure 1-1 below is a simplified schematic of the process. GMI optimised the process design based on the results of several 2009 and 2011 metallurgical testing programs (Blue Coast, 2011; DJB Consultants, 2011; SGS, 2007, 2008a, 2008b, 2009a, 2009b, and 2010) and new metallurgical testwork and analysis for grinding, sedimentation, and filtration in 2014 (Alex G. Doll Consulting Ltd., 2014; ALS Metallurgy Kamloops, 2014; Outotec Canada, 2014a; Outotec Canada, 2014b) and as a result of different analysis (primary crusher technology, stockpile optimisation, tailings filtration plant, waste rock and dry tailings stacking area location, etc.).

The ore will be crushed in a primary gyratory crusher that is located adjacent to the open pit mine and temporarily stored in the coarse ore stockpile. From there it will be conveyed to the processing facilities where it will be ground to 80 percent finer than 106 microns in a semi-autogeneous grinding (SAG) and ball mill circuit.

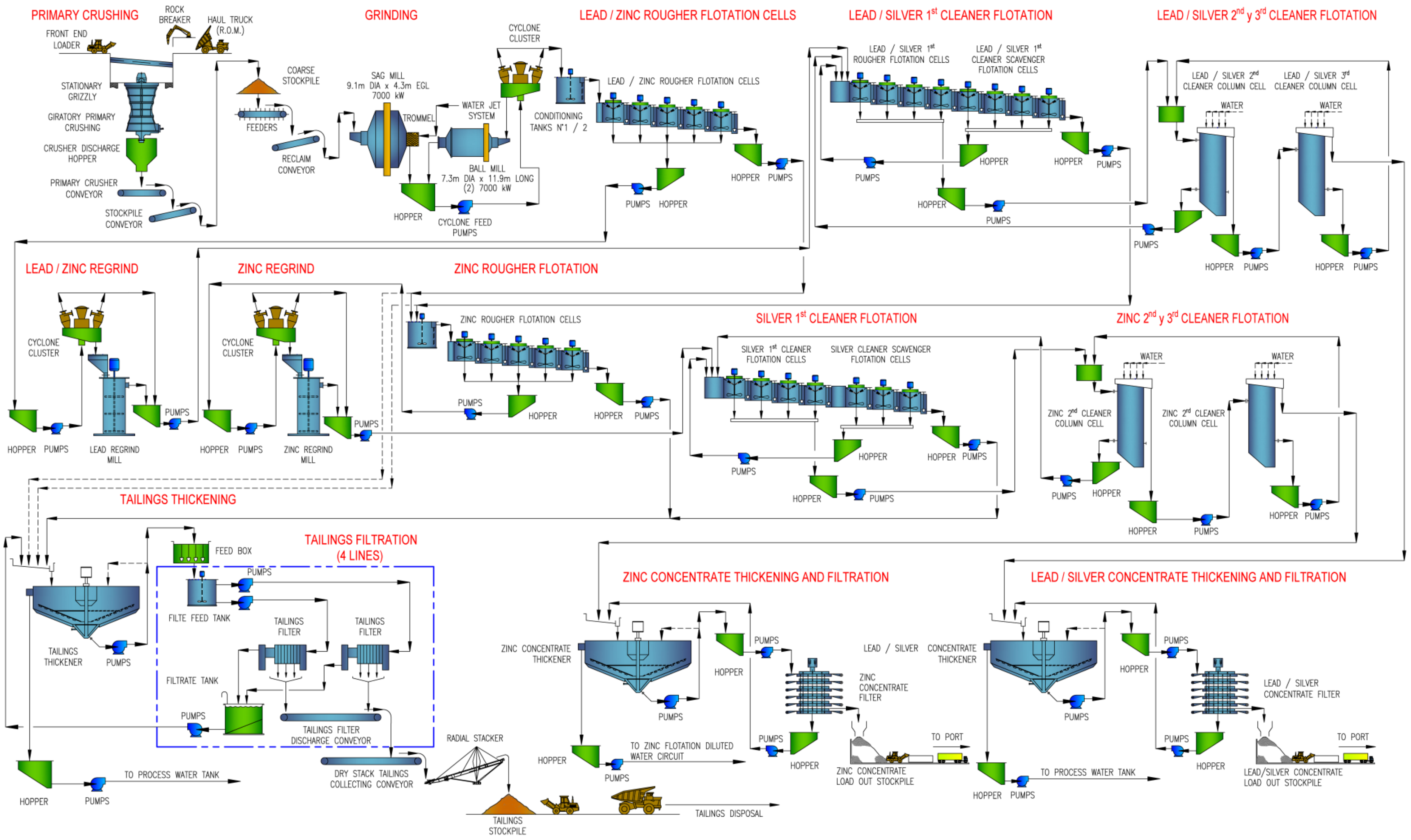
The ore is further processed in a flotation circuit consisting of lead flotation followed by zinc flotation. The majority of the silver will be recovered in the lead flotation circuit and some silver will also be collected in the zinc flotation circuit. Lead sulphide will be recovered in a one-pass rougher flotation bank, producing a concentrate that will be upgraded to smelter specifications in three cleaning stages. Tails from the lead flotation section will then be conditioned for zinc sulphide flotation. The process scheme for zinc flotation also includes a rougher bank and three stages of cleaning using both mechanical cells and column flotation to produce smelter-grade zinc concentrates. For both lead and zinc sections, the rougher flotation concentrates will be reground to 80 percent finer than 25 microns prior to cleaner flotation to liberate the sulphides for further upgrading.

Tailings from the lead and zinc flotation circuit will be thickened, filtered and conveyed to a stockpile at the plant. From there, the filtered tailings will be trucked to the Main Waste Dump where it will be co-disposed with mine waste during the first eight years of operation. Between years 9 and 11, filtered tailings will be disposed of as backfill into the Corani Este pit with additional waste rock. From year 12 onwards, the tailings will be again re-disposed at DDMRP up to the end of the mining operation.

Water will be reclaimed from the tailing thickener overflow and from the tailing filtrate. Process make-up water will be pumped from the contact water section and fresh water section of the Plant Water Supply Pond.

Lead and zinc concentrates will be thickened, filtered, and transported by trucks to the Port of Matarani for ocean shipment to smelters.

Figure 1-1: Simplified Process Flow Diagram for the Corani Project



1.15 PROJECT INFRASTRUCTURE

The infrastructure for the Corani Project requires significant development and planning. The site is remote, at high elevation, and a considerable distance from major urban areas. The infrastructure developed for the Project includes transportation, process buildings and related facilities, water supply and management, power supply, communications, and material storage stockpiles.

Several Project components were optimized subsequent to the July 2015 Technical Report by M3 Engineering (M3 PN 140135, 17 July 2015). Detailed engineering studies, site investigation work, and laboratory testing programs were ongoing at the time of the July 2015 report. The optimizations were presented in December 2015. Several significant changes have been made in the design approach in order streamline the Project and minimize Capital and Operating costs. The most significant changes have been advanced through additional fieldwork and detailed engineering to support the optimization concepts presented in this study. A summary of the infrastructure related work performed subsequent to the 2015 report for the present 2017 study is presented below:

1. More precise and geo-referenced topography completed
2. Soil mechanics studies completed
3. Bridge and tunnel designs for the main Project access road reviewed and in some cases revised
4. Review and optimization of the overall Project layout
5. Review and update of Project hydrology and hydrogeology
6. Evaluation and development of Project accesses
7. Development of accesses for mine vehicles (Haul Roads)

1.15.1 Transportation, Access, and Site Roads

Site Buildings and Facilities

- Mine Services Facilities
- Administration Facilities
- Process Facilities
- Camp Facilities
- Water Supply and Management Facilities
- Power Supply and Distribution
- Communications Systems
- Waste Disposal Facilities
- Waste Rock and Tailings Management Facilities

Transportation

Transportation to and around the site is by roadways that have been developed and improved to accommodate the demands of the Project. An access road has been designed to link the Project site to the Interoceanic Highway that provides access to the town of Macusani and to the rest of the country for receiving supplies and delivering products. The lead and zinc concentrate produced by the mining and mineral processing operations will be delivered to the Port of Matarani or other destination via trucks using the access road and Peru's public highway system.

The current components and arrangement of facilities is described in this section of the Report. Several of these Project components are described in more detail in other sections of this report, and only a general description of the relevant aspects of the Project infrastructure-related components is given in this Section.

Access Road

The Main Access Road to the Corani Project, as designed by GMI and Anddes Asociados SAC (Anddes), will be a new 44 km highway connecting to the existing Interoceanic Highway (34B) which is in turn connected through the existing Peruvian highway system to the Port of Matarani, 632 km. from the Project site. The Port has facilities for concentrate shipment. The Interoceanic Highway is a two-lane, paved highway that connects the Peruvian port cities of Matarani and Ilo.

Buildings and Facilities

Corani Project buildings and facilities are divided into four functional areas: Administration, Mine Facilities, Process Facilities, and Residential Facilities.

The mine facilities include the following:

- Powder magazine 14 m^2 1,800 kg
- Detonator Magazine 18 m^2 500 kg
- Covered ammonium nitrate storage and charging silo 375 m^2
- Emulsion storage silos 2 @ 80 t
- Yard storage
- Truck wash
- Truck fuel storage and
- Tire shop

The administrative facilities are located near the main entrance and include the following:

- Guard house and weigh station
- Administration building
    - Warehouse
    - Fuel supply

The process facility and ancillary buildings include the following:

- Primary crusher
- Crushed Ore Storage and Reclaim
- Grinding, flotation, and reagents buildings
- Concentrate handling and load out building
- Tailings thicker
- Tailings filtration building
- Tailings stacker and stockpile
- Plant maintenance and emergency services
- Analytical laboratory
- Plant water storage and treatment area
- Electrical substation
- Plant water supply pond
- Plant waste water treatment facility

  - Tailings emergency containment pond
  - Drum storage building
- Camp

The residential camp facilities are located approximately 12 km northeast of the mine entrance along the access road and include the following.

  - Dining and food preparation
  - Residential accommodations
  - Medical services
  - Recreational facilities
  - Water and wastewater management
  - Security services

1.15.2 Mine Service Facilities

- Truck and Facility Workshop
- Fleet Management System

1.15.3 Administration Facilities

The administration facilities are located near the main entrance partially because of space constraints and partially to keep suppliers and non-essential personnel out of the mining and process areas. The administration facilities include the main gate and guard house, an administration office building, and a warehouse to receive parts and supplies necessary for operation and maintenance.

- Security
- Warehouse Building
- Administration Building

1.15.4 Project Water Management

Surface water and groundwater will be used to provide the water required for the Project. Surface water (runoff and streamflow) and groundwater (from pit dewatering) will both come from the watershed that hosts the Project. No cross-basin abstractions will be required. . Water on the project is classified as either contact water or non-contact water. Contact water is defined as water that has had contact with any area disturbed by the Project where the water quality could be degraded from Acid Rock Drainage (ARD) or other water contaminants. Non-contact water is defined as water that has not had contact with the Process or any area that has been disturbed. Contact water and non-contact water will be managed and conveyed separately. They will ultimately be stored in a water storage pond which has two separate compartments, one for each circuit. The contact water that has been stored will be consumed as preferential process water (make-up) for the plant. This water cannot be discharged to the environment during operations (See Section 20). A portion of the Non-Contact Water stored in the pond will be used to satisfy the process water demand once the Contact Water has been exhausted. Non-contact Water that is not used will be discharged, if necessary, to the Quebrada Chacaconiza. The Project is required to discharge a fixed quantity of non-contact water downstream as part of the Environmental Impact study and ITS (see Section 20).

1.15.5 Power Supply

A new 138 kV power transmission line is necessary to provide power to the Corani Project. A new power substation will connect with Power Transmission Line L-1013 (San Gabán II – San Rafael – Azángaro) as the power source. A new 138 kV power transmission line will be built to connect the Antapata substation to the Main Corani substation to be built near the Project's main process buildings. The proposed alignment for the 138kV line was provided by Promotora (2015). The transmission line route was selected based on using the route already provided by the Project's Mine Access Road.

1.15.6 Waste Rock and Tailings Management Facilities

Disposal of mine waste and filtered tailings will be in a common deposit, the size of which has been designed for the quantities considered in the mine plan. The height could easily be increased to give more capacity in the future if required. The Main Dump and pits are designed to minimize and mitigate the formation of ARD. ARD is a natural process which arises from the oxidation of sulfide minerals. This risk is present in Corani Waste Rock, tailings, and pit walls. Section 20 describes the ARD management plan.

For the main dump, the primary repository for all mine waste, initially, a base platform will be constructed using Non-Acid Generating (NAG) waste from the mine pre-stripping stages. Filtered tailings and mine waste will be placed in alternating layers, with the waste layers acting as conduits to remove water generated through the tailings consolidation process. Vertical chimneys of rock will connect each layer to increase effectiveness of the drainage system. Drainage water will be directed to a sump located immediately downstream of the deposit. The tailings / waste layers will be encapsulated with NAG waste (10m below, 20m on the down-stream face and 1m on top). Starting in year 9, for 3 to 4 years, a portion of filtered tailings and mine waste will be used to backfill the East pit. Following completion of mining, as part of the closure plan, approximately 24Mt of waste and tailings will be re-handled from the top of the Main Mine Waste and Filtered Tailings Deposit to backfill Minas and Main Pits to the current level of the wetlands.

1.16 MARKET STUDIES AND CONTRACT

Penfold Limited conducted a review of the lead and zinc concentrate markets, smelting charges, penalties, concentrate handling, and land and ocean transportation costs. The supplied information was used as a guide to develop all associated payments and expenses associated with the sale of Corani concentrates. There are no letters of intent or sale agreements in place. All information is based on Penfold's experience for similar concentrates.

1.17 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The main environmental approval required in order to begin mining activities is an Environmental and Social Impact Assessment (ESIA). In 2013, the Ministry of Energy and Mines approved the ESIA based on the Feasibility Study prepared in 2011. The Closure Plan was approved in April, 2015. In 2016, the Ministry of Energy and Mines approved the modification of the ESIA based on the Optimized and Final Feasibility Study prepared by 2015.

The design and operating improvements incorporated in this Technical Report are expected to require only a modification of the existing approved ESIA, without the necessity for additional public hearings, as they are entirely located within the previously approved project footprint. Furthermore, as the environmental impact of the proposed Corani operation has been reduced as a result of the modifications described within this report and the Company anticipates final permitting timelines will shorten and costs will be lower than previously anticipated.

Bear Creek's plans for the Corani project are to focus on preparing for development of the project starting with the preparation and submission of an amended ESIA in the third quarter of 2017.

The community consultation undertaken with the Chacaconiza and Quelcaya communities to date has included a proposal for mining employment, generating widespread acceptance, mainly among younger community members, the teachers at local educational facilities, and community leaders. The current labor force is generally unskilled, mainly working on highway remediation and maintenance. A technical training program is planned to develop the skills of community members to fulfil employment requirements of the Project.

Additionally, Bear Creek completed a Life of Mine ("LOM") Investment Agreement in June 2013. This agreement was entered into with the District of Corani, five surrounding communities, and relevant, ancillary organizations specifying investment commitments over the 23 year project life, including the preproduction period. Under the agreement, annual payments are to be made into a trust designed to fund community projects totaling 4 million nuevos soles per year (approximately $1.6 million per year), beginning with the first installments payable in 2013. Payments will remain constant throughout the pre-development phase and during production. Cessation or interruptions of operations will cause a pro-rata decrease in the annual disbursements. As an integral part of the LOM agreement, a trust or foundation structure is established for approval of investments and disbursement of funds.

1.18 CAPITAL AND OPERATING COST

Life of mine capital cost, capital and life of mine operating cost estimates are summarised in Table 1-5, Table 1-6 and Table 1-7 respectively.

Table 1-5: Life of Mine Capital Cost Summary

Cost Type	Cost (USD Million)
Sunk Costs	4.279
Initial CAPEX	580.919
Sustaining CAPEX	0.361
TOTAL	**585.559**

Table 1-6: Capital Cost Summary

Cost Area	Cost (USD Million)
General	40.337
Mining	44.544
Infrastructure	56.813
Process Plant	245.059
Fresh Water/Process Water	41.549
Power Supply	2.934
Ancillary Buildings	22.977
Engineering	10.000
Project Management & Supervision	14.980
Commissioning, Start Up and Vendor Representatives	13.888
Owner Costs	32.259
Contingency & Escalation	55.579
TOTAL	**580.919**

Table 1-7: Life of Mine Operating Cost Summary

Cost Item	LOM Cost (USD)
Mining	802,268,628
Processing Plant	1,406,214,462
Treatment & Refining Charges	876,771,423
General & Administrative	235,595,236
TOTAL	**3,320,849,749**

1.19 ECONOMIC ANALYSIS

The economic analysis was performed using a Discounted Cash Flow (DCF) which is a standard industry practice. The key assumptions used for the study are shown in Table 1-8 and establish a "Base Case." The table provided the life-of-project averages for grade recovery and these values vary over the life of the project depending on the head grades and split between mixed sulfide ore and transition ore.

Table 1-8: Key Assumptions for the Corani Project – Base Case

Annual ore production – years 1 to end of life (ktonnes)	7,875
Overall process recovery – silver – into both lead and zinc cons	69.9%
Overall process recovery – lead – into lead cons	61.1%
Overall process recovery – zinc – into zinc cons	67.1%
Total processed ktonnes	139,073
Average silver grade (g/t)	50.3 g/t
Average lead grade (%)	0.90%
Average zinc grade (%)	0.59%
Payable ounces of silver net of smelter payment terms (total)	144 million
Payable pounds of lead net of smelter payment terms (total)	1.59 billion
Payable pounds of zinc net of smelter payment terms (total)	1.03 billion
Overall stripping ratio	1.49 to 1
Life-of-Mine years	18

The results of the economic analysis for the Project indicate an after-tax internal rate of return (IRR) of 15.1% and net present value (NPV) of $404.5 million at a 5% discount rate based on metal prices of $18.00 per ounce silver, $0.95 per pound for lead, and $1.10 per pound zinc.

1.20 ADJACENT PROPERTIES

There are no adjacent mineral properties which might materially affect the interpretation or evaluation of the mineralization or exploration targets of the Corani Project.

1.21 OTHER RELEVANT DATA AND INFORMATION

Bear Creek and GMI have created a Project Execution Plan and have conducted a Hazard and Operability study, creating a project development pathway designed to minimize risk and uncertainty, manage construction performance and schedule, to deliver the Project on budget.

The Project is planned to be constructed over a three year time span with engineering continuing through 2017, access road, plant equipment procurement and fabrication beginning the first quarter of 2018. Commissioning and start up is scheduled for the first quarter of 2021.

1.21.1 Objectives

The Project would be executed in accordance with the Execution Plan which is designed to achieve the following objectives:

- Uncompromised safety
- Environmental compliance
- Community relations objectives
- Conformance to the budget
- On-schedule completion
- Compliance with Project quality standards
- Inclusion of Peruvian participation
- Client Satisfaction

1.21.2 Project Development Schedule

The Project development schedule continues with engineering and permitting through the remainder of 2017, followed by road construction and long lead equipment procurement beginning the first quarter of 2018. Construction activities continue through 2019 and 2020 with planned commissioning and startup the first quarter of 2021. GMI has developed the following Project development schedule.

The following list shows the estimate time duration (ETD) for each main activity:

- Detailed Engineering – 15 months
- Permitting – 16 months
- Major Offsite Contracts (Camp, Power Line, Access Road) – 14 months
- Mine Construction/Pre-stripping – 12 months
- Plant Construction – 21 months
- Commissioning and Start-Up – 6 months

The total time from receiving financing to start-up is estimated to be approximately 36 months.

The critical path method should be applied to develop the execution schedule and Primavera P6 software.

1.21.3 Project Management

An experienced EPC company would be selected to develop the Project. This company would develop and implement the Project Procedures Manual that would include the following information:

- Project Management Plan;
- Engineering Management Plan;
- Procurement Plan;
- Logistics and Transportation Plan;
- Construction Plan;
- Commissioning and Startup Plan;
- Quality Assurance Plan;
- Environmental, Health and Safety Plan;

- Communication Plan;
- Project Controls Plan;
- Project Schedule; and
- Project Close-Out Plan.

1.21.4 Engineering

An Engineering Management Plan should be applied which outlines the procedures and tools which would be used to effectively manage the design.

The Project engineering would be developed in two-phases:

- Integration Engineering phase that would confirm and integrate the engineering packages and initiate the procurement of long-lead equipment items,
- Phase 2 Detailed Engineering phase that would be carried out by a leading engineering and construction company following the completion of the Phase 1 Detailed Engineering. As detail engineering designs and quantity take-offs are completed these would be transferred to the procurement and contracts groups for purchase and contracting and to the construction team at the Project site.

The Detailed Engineering work would be developed by an EPC/EPCM company. Some design packages, such as roads and power supply could be executed by a Peruvian company.

1.21.5 Procurement and Contracting

Due to the location and altitude of the site, pre-fabrication and skid-mounted packages would be considered to the greatest extent possible. Pre-fabricated modules would be equipped with piping and valves, wiring and instrumentation to reduce onsite labour.

Sourcing of the majority of equipment and materials is expected to be from USA, Canada, Europe, Chile and China. Some major and minor mechanical equipment and material would be procured from Peruvian suppliers such as platework and steel structure.

Working with the Project construction management team a detailed contracting plan indicating scope breakdown and contract type will be developed during the Project detail engineering phase.

1.21.6 Construction

A Construction Management Plan should be applied which defines the standard processes that must be used to construct the Corani Project in a controlled and efficient manner.

The Project scope will be divided into manageable work areas to facilitate a controlled work flow and smooth handover from construction to commissioning through to production.

The Project construction team is responsible for the delivery of the works in accordance with the agreed scope of work and the Project schedule.

Specific timing for all engineering work packages and construction ERFP packages would be included in the Project master schedule.

1.21.7 Commissioning and Startup

A Commissioning Management Plan will be applied which defines the standard processes that must be used to commission the Corani Project in a controlled and efficient manner.

Commissioning includes those activities necessary for an effective transition between construction and mechanical completion when systems are turned over to the commissioning and start-up team. These activities include the following:

- Ensure that equipment is operationally ready for start-up (i.e. to accept feed);
- Sequence starting and running of tested logical groups of equipment;
- Wet and dry runs of systems;
- Demonstration of the suitability of the facilities to be ready for processing and production; and
- Coordinate with and assist the owner to achieve hand over of the completed facilities.

The commissioning and start up team is planned to be an integrated organization of plant operators, contractors and suppliers.

1.22 RECOMMENDATIONS

The recommendations provided below address areas that require more complete definition to inform and optimize the detailed engineering design.

1.22.1 Geotechnical

A monitoring plan, monitoring instrument alert levels and operation and contingency manual, should be developed for the whole mine facilities. A best industry practice should be applied in the development of those documents, mainly for the critical components. Those documents and manuals will support the routine operation and will produce a safe environment.

A group of specialists, including geotechnical, hydrologist and mechanical engineers, should carry out Annual Safety Reviews of the whole facilities, taking into consideration the Canadian Dam Association guidelines. The Safety Report should update the monitoring plan, operation and contingency manual, stability condition, water management, critical equipment operation of each mine facility and provide action lists for improving the functioning of them. Regardless the Annual Safety Review, all the facilities will have to be inspected after the occurrence of an extreme storm or earthquake event.

Due to the geotechnical characteristics of the Plant Water Pond foundation, a good quality control and quality assurance program should be implemented during construction, including geoelectrical surveying for verifying potential geomembrane defects which produce leaks.

Additional geotechnical drilling should be completed within the planned pit. This will confirm the current pit slope design basis and potentially allow an increase in the pit slope angles. The pit will intersect the unconsolidated sediments lining the floor of the upper bofedal and lower bofedal areas. Additional drilling, testing, and analyses are required to design the pit slopes within the bofedal soils and to develop a detailed plan for dewatering and mining the bofedal soil material. This will involve drilling several boreholes through the unconsolidated sediments, with production of detailed stratigraphic logs and undisturbed sampling for density and strength testing of the unconsolidated material. Boreholes would be completed as monitoring wells, and multiple-well aquifer testing will be performed to better assess the dewatering requirements for the material.

1.22.2 Process Plant Design

During the initial stages of the next design phase, the process plant design should be optimised considering the last mining plan.

Confirm the burden and spacing of blasting in order to reduce the stationary grizzly oversize (1 m opening) and the rock breaker duty.

It is recommended to optimise the feed and discharge primary crusher bins capacities in order to adjust to the haul trucksize chosen.

Also it should be evaluated from the environmental point of view if a cover for the coarse ore stockpile is required.

Further work will be required during detailed design on the pebbles handling system, once more data is obtained on their effects on SAG mill operation.

It is suggested to incorporate metallurgical samplers in lead and zinc flotation feed circuits and both final concentrates and tailings in order to be able to realize auditable mass balances for silver, lead and zinc.

Also it is recommended to carry out new tests using SFR flotation cells, with the purpose of reducing the amount of penalty elements (As, Sb, clays) and maximising the silver recovery in the lead concentrate.

It is unknown the effect that the water recirculation will have on the lead flotation efficiency, therefore, with the aim of guaranteeing appropriated lead and silver recovery it is suggested to use two independent process water circuits for lead and zinc. This could mean a reduction in reagents consumption, thus saving operating costs for the process plant.

It is also suggested to incorporate into the process plant design the in-line addition of hydrogen peroxide and activated carbon.

For pH control of the recirculated water it may be required to design a system for sulphuric acid addition and adjust the pH prior to lead flotation.

Finally, it is recommended to carry out a dynamic simulation in order to detect possible bottle necks and optimise the design of the plant. This should be done during the initial stages of the next design phase.

1.22.3 Environmental Studies, Permitting and Social or Community Impact

In 2016, the Ministry of Energy and Mines approved the modification of the ESIA based on the Optimized and Final Feasibility Study prepared by 2015.However, It is recommended that Bear Creek commence the permitting process on water rights and mine plan approval, both of which are critical, early-stage permits. Bear Creek is also encouraged to utilize all efforts in maintaining its social license and ensuring the continued strong support from local communities, local and regional governments, and the central Peruvian government.

Additional evaluation is required on the water balance. The main dump seepage model and the pit dewatering model both require a higher level of rigor to achieve the level of “detailed design”. Once these studies are complete, the water balance will require revision. However, the water balance is robust (see Section 20) and few complications are foreseen.

1.22.4 Capital and Operating Cost

It may be possible to reduce the cost of delivering tailings to the Main Dump and pit backfill disposal sites by varying the proportion of tailings delivered by conveyor systems and by trucks during the period when tailings are being produced. It is recommended that an optimization study be carried out to determine this, and a detailed plan should be devised. Tailings will be co-disposed with waste rock in the Main Dump and pit backfills. In general, it is expected that it will be cheaper to use conveyors instead of trucks to deliver tailings to ultimate disposal destinations, but exclusive use of conveyors may be less practical for tailings destined for the pit backfill. During the period when pit backfilling will be taking place, the current schedule indicates that truck capacity will be available; therefore, an optimization study should specify the ideal mix of conveyor/truck transport of tailings over time, depending partly on truck availability.

A study should be conducted to match operating equipment to the high-altitude conditions, potentially identifying equipment outfitted with pressurized cabs and other worker comfort and performance additions. Caterpillar equipment offers high-altitude arrangements (HAA), and these modifications allow their power ratings to be valid to 4,877 m.

1.23 Risks

The following risks have been identified:

- The high altitude of the site may have a greater-than-expected negative impact on worker productivity.
- The high altitude of the site may result in greater-than-expected impacts on the function and capacity of diesel-powered equipment and electrical components.
- As with any large-scale mine development, there is a risk that additional capital may be difficult to raise in the event that costs increase during the pre-production period.
- A currency exchange risk exists. While a weakening of the Peruvian Nuevo Sol (PEN) would lower the cost of in-country expenses, conversely, strengthen of the PEN would increase local cost.
- Although local communities have generally supported the Project development, there is a risk that sentiments could change, or that special interest groups from outside the community could mobilize opposition to the Project.
- During operations, a potential silver migration from the lead flotation circuit to zinc flotation circuit must be evaluated with additional metallurgical test.

1.24 Opportunities

The following opportunities have been identified:

- It may be possible to improve metal recoveries by optimizing flotation work. Testing under optimized conditions could increase recovery over that predicted by the geometallurgical model.
- Data generated during additional geotechnical drilling may show that it is feasible to steepen pit slopes.
- It may be possible to improve concentrate grades and increase the net smelter return.
- Operating cost improvements may be obtained from using conveyor systems to transport tailings to the disposal sites.
- Complete further test work to optimise the recovery of Silver in the Lead Flotation Circuit, potentially increasing the Project revenue.
- Optimise the plant design by developing a dynamic simulation model.
- Gold Zone: The Gold Zone is an advanced exploration target on the Corani Property. To advance the Gold Zone target, additional metallurgical test work should be undertaken to identify an appropriate recovery method so that capital and operating costs can be developed for a recovery plant. When a recovery strategy is determined, a mineral resource should be estimated for this area so that scoping level studies can be undertaken to evaluate the Gold Zone's economic potential. (1)

  (1): Recommendation from the 2015 Ni43-101 report

2 Introduction

Bear Creek Mining Corporation is a leading Peru-focused silver exploration and development company listed as BCM on TSX Venture and BCEKF on OTC. They are focused on developing their Corani Project located in the Department of Puno in southern Peru.

2.1 Purpose

Bear Creek Mining Corporation (BCM) has requested Sedgman to coordinate the update of the technical report National Instrument 43-101, developed in 2015 by M3. Between December 2016 and July 2017 GMI developed the "Phase I - Detail Engineering" of the Corani Project, which was based mainly on the Feasibility Study Updated in 2015 by M3 Engineering & Technology Corp. (M3). Based on the above, this report was prepared in accordance with "Form 43-101F1, Technical Report" of the Canadian Securities Administrators National Instrument 43-101 (NI 43-101).

The Project includes crushing, grinding and flotation of mixed sulphides, and transitional ores, for proven and probable Mineral Reserves containing 139 million tonnes of ore having a Life-of-Mine average grade of 50.3 grams per tonne of silver, 0.90% lead, and 0.59% zinc, containing, 225 million ounces of silver, 2.7 billion pounds of lead and 1.8 billion pounds of zinc over a period of 18 years of mining.

The completed engineering (FEED) by BCM is the first step to advance with the engineering details of the Project in order to confirm the estimation of investment, add value to the designs proposed by the feasibility engineering, and develop new or with more detailed engineering for:

- Optimization of the new mining plan
- Review of metallurgical test work of Project
- Geo referenced topography and greater precision
- Studies of Soil Mechanics
- Review of bridge and tunnel design in the Main Project Access
- Review and update of hydrology of the Project
- Development of designs for tailings filter area,
- Development of access for mine vehicles (Haul Roads)
- Trade off development between Jaw Crusher and Gyratory crusher
- Trade off development of transport between conveyor and truck mine
- Update local construction cost
- Get offers for the top teams and select the top suppliers.
- Development of designs which allow for confirmation of quantities for items with high cost incidence
- Update of Capex and Operating Cost with precision range of (±15%)
- Update of the Project Development Schedule and Project Execution Plan

The reserves of the present study are practically the same as the reserves described in report NI 43-101 (M3 Engineering, 2015). The reserve tonnage is 1% higher in the Study and the silver and lead metal is 1% lower, and zinc metal is approximately 1.4% higher.

During the pre-strip period, 99.8% of the waste to be removed will be NAG, and will be used to construct initial mine haul roads, the primary crusher platform and the base of the Main Mine Waste and filtered tailings.

2.2 Sources of Information

This Report is the product of technical contributions from a number of consultants; together with BCM personnel. Listed below are the primary "Qualified Persons" (as defined in the National Instrument 43-101) that compiled different sections of the report.

Table 2-1 describes the primary contributors by section.

- Juan Carlos (JC) Tapia, ChE, IMCh, PE, QP (CRIRSCO) - General Coordinator of the Study, Summary, Recovery Methods, Interpretation, Conclusions and Recommendations.
- Terre Lane, P.E., QP-MMSA – Resource and Reserve Estimation, Mine Engineering, Relevant Data and Economic Analysis,
- Deepak Malhotra, Ph.D., QP-MMSA– Mineral Processing and Metallurgical Testing,
- Dennys Parra, PE – Geotechnical, Water and Environmental Studies
- Jennifer J. (JJ) Brown, P.G., QP-SME-RM, Geological Setting and Mineralization, Deposit Types, Exploration, Drilling, Sample Preparation, Analyses and Security & Data Verification,
- Rick Moritz, P.Eng., Global Resource Engineering Ltd. - Geometallurgy
- Gregory Wortman, BE (Metallurgy) PE. (APEGBC) QP, Project Infrastructure
- Michael Short, BE (Civil), CEng FIMMM, FAusIMM(CP), FIEAust CPEng, Capital and Operating Costs
- J. Larry Breckenridge, Environmental Studies, Geochemistry, Permitting and Social or Community Impact, Water supply subsection, PE (Main Dump & Pit Backfill Geotech - Anddes)

Table 2-1: List of Contributing Authors

SECTION	SECTION NAME	Main Contributor	Qualified Person
1	Summary	Sedgman	Juan Carlos (JC) Tapia ChE, IMCh, PE (CRIRSCO) QP
2	Introduction	Sedgman	Juan Carlos (JC) Tapia ChE, IMCh, PE (CRIRSCO) QP
3	Reliance on Other Experts	Sedgman	Juan Carlos (JC) Tapia ChE, IMCh, PE (CRIRSCO) QP
4	Property Description and Location	GRE	Kevin Gunesch - P.E.
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	GRE	Kevin Gunesch - P.E.
6	History	GRE	Kevin Gunesch - P.E.
7	Geological Setting and Mineralization	GRE	Jennifer J. (JJ) Brown P.G., QP-SME-RM
8	Deposit Types	GRE	Jennifer J. (JJ) Brown P.G., QP-SME-RM
9	Exploration	GRE	Jennifer J. (JJ) Brown P.G., QP-SME-RM
10	Drilling	GRE	Jennifer J. (JJ) Brown P.G., QP-SME-RM
11	Sample Preparation, Analyses and Security	GRE	Jennifer J. (JJ) Brown P.G., QP-SME-RM
12	Data Verification	GRE	Jennifer J. (JJ) Brown P.G., QP-SME-RM
13	Mineral Processing and Metallurgical Testing	GRE GRE	Deepak Malhotra - Ph.D., QP-SME-RM Rick Mortiz - QP-MMSA
14	Mineral Resource Estimates	GRE	Terre Lane - QP-MMSA
15	Mineral Reserve Estimates	GRE Anddes	Terre Lane - QP-MMSA Denys Parra – P.E
16	Mining Methods	GRE GRE Anddes	Terre Lane, QP-MMSA Kevin Gunesch - P.E. Denys Parra – P.E
17	Recovery Methods	Sedgman	Juan Carlos (JC) Tapia ChE, IMCh, PE (CRIRSCO) QP
18	Project Infrastructure	Sedgman GRE	Gregory Wortman BE (Metallurgy) PE. (APEGBC) QP J. Larry Breckenridge - P.E
19	Market Studies and Contracts	GRE	Kevin Gunesch – P.E.
20	Environmental Studies, Permitting and Social or Community Impact	GRE Anddes	J. Larry Breckenridge - P.E Denys Parra - P.E
21	Capital and Operating Costs	GBM	Michael Short BE (Civil), CEng FIMMM, FAusIMM(CP), FIEAust CPEng
22	Economic Analysis	GRE	Terre Lane, QP-MMSA
23	Adjacent Properties	GRE	Jennifer J. (JJ) Brown P.G., QP-SME-RM
24	Other Relevant Data and Information	GRE	Terre Lane - QP-MMSA
25	Interpretation and Conclusions	Sedgman	Juan Carlos (JC) Tapia ChE, IMCh, PE (CRIRSCO) QP
26	Recommendations	Sedgman	Juan Carlos (JC) Tapia ChE, IMCh, PE (CRIRSCO) QP
27	References	Sedgman	Juan Carlos (JC) Tapia ChE, IMCh, PE (CRIRSCO) QP

Abbreviations: ALL – All QP Contributors; BCM – Bear Creek Mining Corporation; GRE – Global Resource Engineering Ltd; GBM Minerals Engineering Consultants Limited – Global Engineering Mining; Anddes – Anddes Asociados SAC

Note: Where multiple authors are cited, refer to author certificate (Appendix A) for specific responsibilities.

This Report has been compiled for BCM by Sedgman, GRE, ANDDES and GBM. The Report is based on information and data supplied to the Authors mainly by BCM, and other parties. The Authors have relied in this information and on information provided from previous studies.

Where possible, the Authors have confirmed the information provided by comparison against other data sources, similar projects in Peru and South America or by field verification where checks and confirmations were not possible. Where such estimates and approximations have been used, it is so noted and the assumptions made in making such estimates and approximations are so noted.

The Authors have reviewed and incorporated that all information supplied in the previous technical report is complete, and reliable, within normally accepted limits of error. During the normal course of the review, the Authors have not discovered any reason to doubt that assumption.

This Report conforms to the standards of a NI43-101 Technical Report, and is based on the work completed to date for the Phase I - Detail Engineering of the Corani Project, and to evaluate the economics of the plant that will operating at 22,500 MTPD as an annual average.

The study sets forth conclusions and recommendations, based on the Authors' experience and professional opinion, which result from their analysis of work and data collected.

This Report should be construed in light of the methodology, procedures and techniques used for its preparation, and should be read in original context - all readers should refer to referenced documents for clarification of the original context.

2.3 Site Visit & Personal Inspections

The following site visits were made by the groups and individuals listed below.

- Jennifer J. (JJ) Brown visited the site from August 7 to 9, 2017
- Larry Breckenridge, P.E. of Global Resource Engineering Ltd. visited the site on five occasions from May 2011 through April 2012.
- Kevin Gunesch has visited the site on 4 separate occasions in 2011

The review coordinator Juan Carlos Tapia visited the site in December 2016, at the beginning of the Phase I - Detail Engineering

The following specialists have not visited the site and are not physically aware of the Project site:

- Terre Lane and Rick Moritz of Global Resource Engineering Ltd.
- Gregory Wortman of Project Infrastructure
- Michael Short of Capital and Operating Cost

2.4 Terms of Reference

The units of production in this report are metric unless otherwise noted. Production is in tonnes (t). All monetary amounts are in May 2017 US dollars along with other variables such as the price of silver, lead and zinc, unless otherwise noted.

Table 2-2: List of Acronyms

Acronym	Definition
AA	Atomic absorption
ABA	Acid-Base Accounting
Ag	Chemical symbol for silver
ANA	Water National Authority
AP	Acid (generating) Potential
ARD	Acid Rock Drainage
As	Chemical symbol for Arsenic
BCM	Bear Creek Mining Corporation
bcm	Bank Cubic Meter
Bi	Chemical symbol for Bismuth
CAPEX	Capital Expenditure
CATV	Cable TV Distribution System
CERTAG	Silver Value from Assay Certificate
CFP	Cumulative frequency plot
CIF FO	Cost Insurance & Freight Free Out
CM	Construction Manager
Co	Chemical symbol for Cobalt
CS	coarse-grained silica-sulfide
CSA	Canadian Securities Administrators
CSC	Coarse-grained silica-sulfide-celadonite
Cu	Chemical symbol for Copper
DCS	Distributed Control System
EDO	Emulsified Diesel Oil
EPC	Engineering Procurement and Construction
EPCM	Engineering Procurement and Construction Management
ERFP	Engineering Requisition for Purchase
ESIA	Environmental and Social Impact Assessment
F	Chemical symbol for Fluorine
FBS	Fine-grained black silica-sulfides
Fe	Chemical symbol for Iron
FEED	Front End Engineering Design
FeO	Iron oxide
FS	Feasibility Study
G&T	G&T Metallurgical Services
GA	General Arrangement
GBM	GBM Minerals Engineering Consultants Limited
GFA	General Facilities Arrangement
GMI	Graña Montero Ingeniería
GPS	Global positioning system
GRE	Global Resource Engineering Ltd.
HAZOP	Hazard and Operability Study
HDPE	High density polyethylene
Hg	Chemical symbol for Mercury
ICP	Inductively-coupled plasma
ID2.5	Inverse Distance to the 2.5 Power

Acronym	Definition
ID3	Inverse Distance to the 3rd Power
IDP	Inverse Distance to a Power
IEM	Impuesto Especial a la Minería (Special Mining Tax)
IFC	International Finance guidelines
IGV	Impuesto General a las Ventas (Peruvian value added tax)
IMC	Independent Mining Consultants
INACC	Instituto Nacional de Concesiones y Catastro Minera
INGEMMET	Geologic Mining and Metallurgical Institute (Instituto Geológico Minero y Metalúrgico)
IP	induced polarization
IRA	Inter-ramp angles
IRR	Internal Rate of Return
LCT	Locked Cycle Test
LOM	Life of Mine
M3	M3 Engineering & Technology Corporation
MARC	Maintenance and Repair Contract
MgO	Chemical symbol for Magnesium Oxide
MINEM	Ministerio de Energía y Minas
Mn	Chemical symbol for Manganese
MnO	manganese oxide
NAG	Non Acid Generating
NI43-101	Technical Report of the Canadian Securities Administrators National Instrument 43-101
NNP	Net Neutralization Potential
NP	Neutralization Potential
NPV	Net Present Value
NSR	Net Smelter Return
OEFA	Organismo de Evaluación y Fiscalización (Agency for Environmental Assessment and Enforcement)
OPEX	Operating Expenses
OSINERGMIN	Organismo Supervisor de la Inversión en Energía y Minería (Supervisory Agency for Investment in Energy and Mining)
PAG	Potentially Acid Generating
Pb	chemical symbol for lead
PDS	Power Distribution Center
PE	Plan of Execution
PEA	Preliminary Economic Assessment
PEN	Peruvian New Sol (currency)
Penfold Limited	Concentrate marketing and sales consultant
PEP	Project Execution Plan
PFS	Prefeasibility Study
PG	plumbogummite
PM	pyrite marcasite + quartz
PMT	Post Mineral Tuff
QA/QC	Quality Analysis/Quality Control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy.
QP	Qualified Persons

Acronym	Definition
QSB	crystalline quartz sulfide-barite
R^2	Coefficient of Determination
RDi	Resource Development, Inc. (Wheat Ridge, Colorado)
RF	Revenue Factor (Whittle)
ROI	Return on Investment
SAMPNO	Sample Number
Sb	Chemical symbol for antimony
SEDGMAN	Process Engineering Group
SGS	SGS Mineral Services Laboratory
SOW	Scope of Work
Sph	Spherical
GyM	Mine Engineering Design Firm
T/C	Treatment charge
TET	silver-bearing tetrahedrite
TS	TS Technical Services (Tom Shouldice)
TSF	Tailings Storage Facility
UTM	Universal Transverse Mercator
VDG	VDG del Perú S.A.C.
WRF	Waste Rock Facility
Zn	chemical symbol for zinc

Table 2-3: Glossary

Term	Definition
Bofedal	Organic soil found in the wet areas in the central parts of the valley
Campesino	A term in Spanish meaning farmer.
Quebrada	A term in Spanish, meaning gorge, valley or draw.
Tailing	Finely ground materials from which the desired mineral values have been largely extracted. Typically, approximately 98 per cent of the material mined for processing is discharged as tailing.
Waste rock	Material such as soils, barren or uneconomic mineralized rock that surrounds a mineral ore body and must be removed in order to mine the ore. This is generally referred to as waste rock in metalliferous mines.

Table 2-4: Units of Measure

Unit Abbreviation	Definition
cm	centimetre
d	day
dmt	dry metric tonne
ft	foot
g	gram
g/t	gram per metric tonne (metric), equivalent to Parts Per Million
gm/t	gram per metric tonne (metric), alternate spelling
h	hour
ha	hectare
hp	Horsepower
kg	kilogram

Unit Abbreviation	Definition
kg/t	kilogram per tonne (metric)
km	kilometer
km2	Square kilometers
kph	kilometers per hour
kt	kilotonne
ktonnes	kilotonnes
ktpy	kilotonnes per year
kW	kilowatts
kWh	kilowatt hours
kWh/t	kilowatt hours per tonne (metric)
lb	pound
m	meter
m^2	square meter
m^3	cubic meter
Ma	Million Years
masl	meters above sea level
Mbcm	million bank cubic meters
min	minutes
mm	millimeters
Moz	Million of troy ounces
MPa	million Pascals
Mt	million tonnes
opy	ounces per year
oz	Troy ounce
oz/dmt	ounces per dry metal tonne
Pb	Chemical Symbol for Lead
ppm	parts per million
t	tonne (metric)
tpd	tonnes (metric) per day
tph	tonnes (metric) per hour
t/m3	Tonnes per cubic meter (density)
tpy	tonnes (metric) per year
wmt	wet metric tonne
µm	Micrometer (microns)
$/dmt	dollars per dry metal tonne
$/payable oz	dollars per payable ounce
$/wmt	dollars per wet metal tonne
%	percent

3 Reliance on Other Experts

The Qualified Persons (QPs) for this report have relied on certain reports, opinions and statements of legal and technical experts who are not considered “Qualified Persons”, as defined by NI 43-101. Reports received from other experts have been reviewed for factual errors by the relevant QPs and determined that they conform to industry standards, are professionally sound, and are acceptable for use in this Report. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports.

3.1 Mining Concessions

Legal review concerning the status of mineral concessions covering the Corani Project was conducted by Estudio Grau Abogados. The status report (Estudio Grau, 2015) with respect to the mining concessions on the Project property stated that the claims are in good standing and that BCM owns the title.

3.2 Surface Rights

Elsiario Antunez de Mayolo of BCM provided information regarding ownership of the surface rights for the project in a telephone call with Kevin Gunesch on 25-Oct-2017.

3.3 Taxation and Social Costs

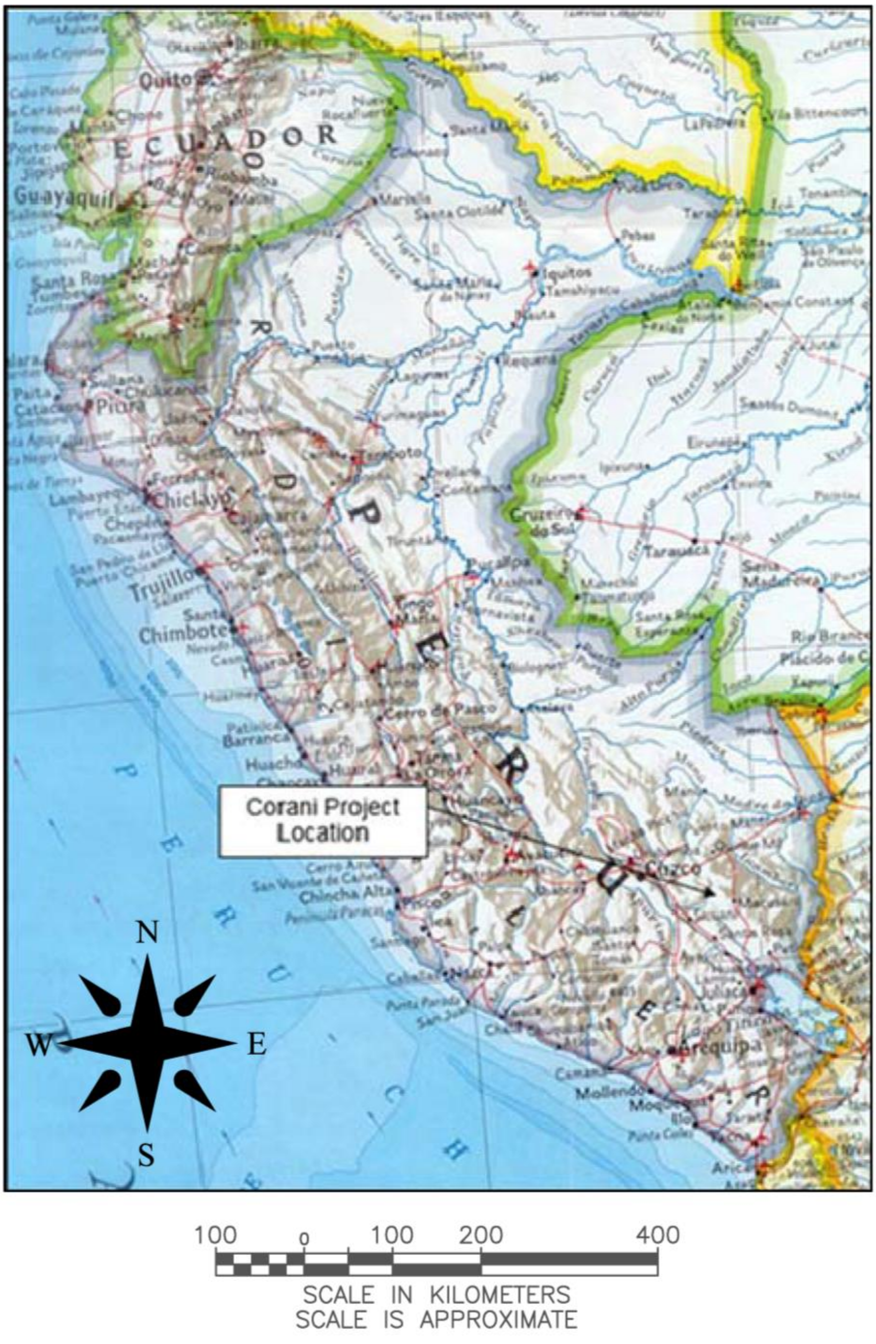
Bear Creek Mining provided all estimates of Peruvian tax code, estimation of taxes, and social costs.

4 Property Description and Location

4.1 Location

The Project site is located in the Andes Mountains of south-eastern Peru at elevations of 4800 to 5100 meters above sea level (masl), specifically within the Cordillera Vilcanota of the Eastern Cordillera. The site is located in the Region of Puno, immediately northeast of the continental divide that separates Pacific drainages from Atlantic drainages. The site location is approximately 160 km in a direct line to the southeast of the major city of Cusco, with Universal Transverse Mercator (UTM) coordinate ranges of 312,000E to 322,000E and 8,443,000N to 8,451,000N. Figure 4-1 illustrates the general location on the map of Peru. The nearest town of significant size and infrastructure is Macusani, which is located around 30 km to the east of the Project.

Figure 4-1: Corani Project Location in Peru



4.2 Mineral Tenure

4.2.1 Summary

The land status of the Project is a series of twelve mineral claims or concessions. Mineral concessions in Peru are filed with the Instituto Nacional de Concesiones y Catastro Minera (INACC) which is part of the Ministerio de Energía y Minas in Peru (MINEM). Claims can vary in size from 100 to 1,000 ha. They are rectangular geometries parallel to the UTM grid system employed in the district. The Corani Project is located in the district of Corani, province of Carabaya department of Puno, in Peru, and covers an aggregate extent of 5,180.1213 hectares.

4.2.2 Purchase Agreements

On March 15, 2007, the Company (BCM or "Bear Creek") and Rio Tinto Mining and Exploration Ltd. ("Rio Tinto") executed a definitive option and shareholders' agreement (the "Option Agreement") in respect of the Corani Property. The Option Agreement formally defines and confirms the terms as set out in the Letter of Understanding signed between the parties on January 19, 2005. Refer also to "Mineral Projects – Corani Silver-Zinc-Lead Property."

On January 15, 2008, the Company made the final US$3 million payment to Rio Tinto under the Option Agreement, resulting in the Company owning 70% of the Corani Property, subject to certain success payments, purchase rights provisions, and claw-back rights as previously disclosed. Under the terms of the Option Agreement, upon Bear Creek earning its 70% interest, the parties were required, within 100 days of January 15, 2008, to enter into a joint venture agreement, with Rio Tinto having a 30% interest, and dilution provisions for each party which conform to industry practices.

On March 6, 2008, Bear Creek entered into an agreement (the "Purchase and Sale Agreement") with Rio Tinto, which was subsequently amended, as described below, to purchase Rio Tinto's remaining 30% interest in the Corani Project and extinguish all of Bear Creek's future payment obligations, royalties, and Rio Tinto's back-in rights under the Option Agreement. Bear Creek agreed to pay Rio Tinto US$45 million and to issue Rio Tinto 3,871,000 common shares as follows: (i) 3,871,000 Bear Creek common shares which were issued on July 16, 2008; (ii) US$5 million in cash payable by December 31, 2008; (iii) US$15 million in cash payable by the earlier of December 31, 2008 or 15 business days following a change of control of the Company; and (iv) US$25.0 million in cash payable on the earlier of December 31, 2009 or 180 days following a change of control of Bear Creek.

On July 17, 2008, the Company amended the terms of the Purchase And Sale Agreement and agreed to issue an additional 120,000 common shares to Rio Tinto, in consideration for which Rio Tinto extended US$15 million of the US$20 million cash payment which has been required to be made under the Purchase And Sale Agreement by the earlier of December 31, 2008 and 15 business days following a change of control of the Company, to the earlier of September 30, 2009 and 90 days following a change of control of the Company. This increased the number of common shares issuable to a total of 3,991,000 shares.

On February 27, 2009, the Company entered into an amendment agreement (the "Amendment Agreement") with Rio Tinto with respect to its purchase of Rio Tinto's remaining 30% interest in the Corani Project. Under the Amendment Agreement, Rio Tinto agreed to restructure the final two cash payments of US$15 million previously due on September 30, 2009 and US$25 million previously due on the earlier of December 31, 2009 or 180 days following a change of control of Bear Creek. In consideration for deferring the majority of these payments out several years, the purchase price increased from US$75 million to US$77.2 million, representing an increase of US$2.2 million, of which US$36.1 million had been already paid in shares or cash. The restructured remaining payments were then:

- US$10 million due 30 September 2011
- US$15 million due 30 June 2012.

Bear Creek agreed to make the following additional payments in consideration for the restructuring payable in either cash or shares, at the option of Bear Creek:

- US$1.1 million upon signing of the Amendment Agreement or as soon thereafter as TSX-V acceptance was received, which was paid on March 11, 2009 by the issuance of 1,021,266 shares of Bear Creek
- US$1.1 million cash, which was paid on January 10, 2011.

Additionally, the Amendment Agreement immediately removed the accelerated payment condition upon change of control of Bear Creek. The Amendment Agreement also provided for the reduction of the security against the balance of the payments, to security charging only the Corani Property upon completion of US$10 million.

On February 3, 2011, the Company entered into an additional amendment agreement (the **"2011 Amendment Agreement"**) whereby Rio Tinto agreed to accept a final payment of US$23 million in lieu of and in full satisfaction of the remaining two cash payments of US$10 million due on September 30, 2011 and US$15 million due on June 30, 2012. Accordingly, the Company has paid the entire obligation for its 100% interest in the Corani project early and received a discount of 8% of the balance of the required payments for doing so.

This final payment extinguished all security interests, share pledges and other encumbrances that Rio Tinto held over the Corani Project and Company's other assets. Copies of the Purchase and Sale Agreement, the Amendment Agreement and the 2011 Amendment Agreement may be obtained under the Company's profile on the SEDAR website (www.sedar.com).

4.2.3 Property Identification

BCM contracted Grau Abogados in Lima Perú to provide a legal opinion regarding the standing of the twelve (12) mineral concessions controlled by the company in 2011 and 2015. According to Estudio Grau (2015), BCM and its subsidiaries own 100% of the title to the twelve (12) mineral concessions comprising the Corani Project, listed in Table 4-1. Control and current status were verified in August 2017 through an electronic database search of the Geologic Mining and Metallurgical Institute (INGEMMET). All concessions are in good standing.

Table 4-1: Mineral Concessions comprising the Corani Project

Identification Code	Name	Holder	Available Hectares[1]	Status	Province	District
10250805	CHAUPITERA	BEAR CREEK MINING S.A.C.	800	Current	CARABAYA	CORANI
10251005	CORANI 100	BEAR CREEK MINING S.A.C.	5[2]	Current	CARABAYA	CORANI
10251105	CORANI 200	BEAR CREEK MINING S.A.C.	21.9730[2]	Current	CARABAYA	CORANI
10068505	CORANI 5	BEAR CREEK MINING COMPANY SUCURSAL DEL PERÚ	93.2601[2]	Current	CARABAYA	CORANI
10289403	CORANI I	BEAR CREEK MINING S.A.C.	300	Current	CARABAYA	CORANI
10289503	CORANI II	BEAR CREEK MINING S.A.C.	300	Current	CARABAYA	CORANI
10021905	CORANI III	BEAR CREEK MINING COMPANY SUCURSAL DEL PERÚ	300.0074	Current	CARABAYA	CORANI
10289203	MINAZPATA 1	BEAR CREEK MINING S.A.C.	1,000	Current	CARABAYA	CORANI
10289303	MINAZPATA 2	BEAR CREEK MINING S.A.C.	300	Current	CARABAYA	CORANI
10038904	MINAZPATA 3	BEAR CREEK MINING S.A.C.	1,000	Current	CARABAYA /MELGAR	CORANI /MACUSANI/NUÑOA

Identification Code	Name	Holder	Available Hectares[1]	Status	Province	District
10357604	MINAZPATA 4	BEAR CREEK MINING S.A.C.	159.8808[2]	Current	CARABAYA	CORANI
10250905	PACUSANI	BEAR CREEK MINING S.A.C.	900	Current	CARABAYA	CORANI
	All	All	5,180.1213			

Obtained from: http://www.ingemmet.gob.pe
[1] Available area not including overlaps with prior mineral concessions
[2] Overlaps with existing concession

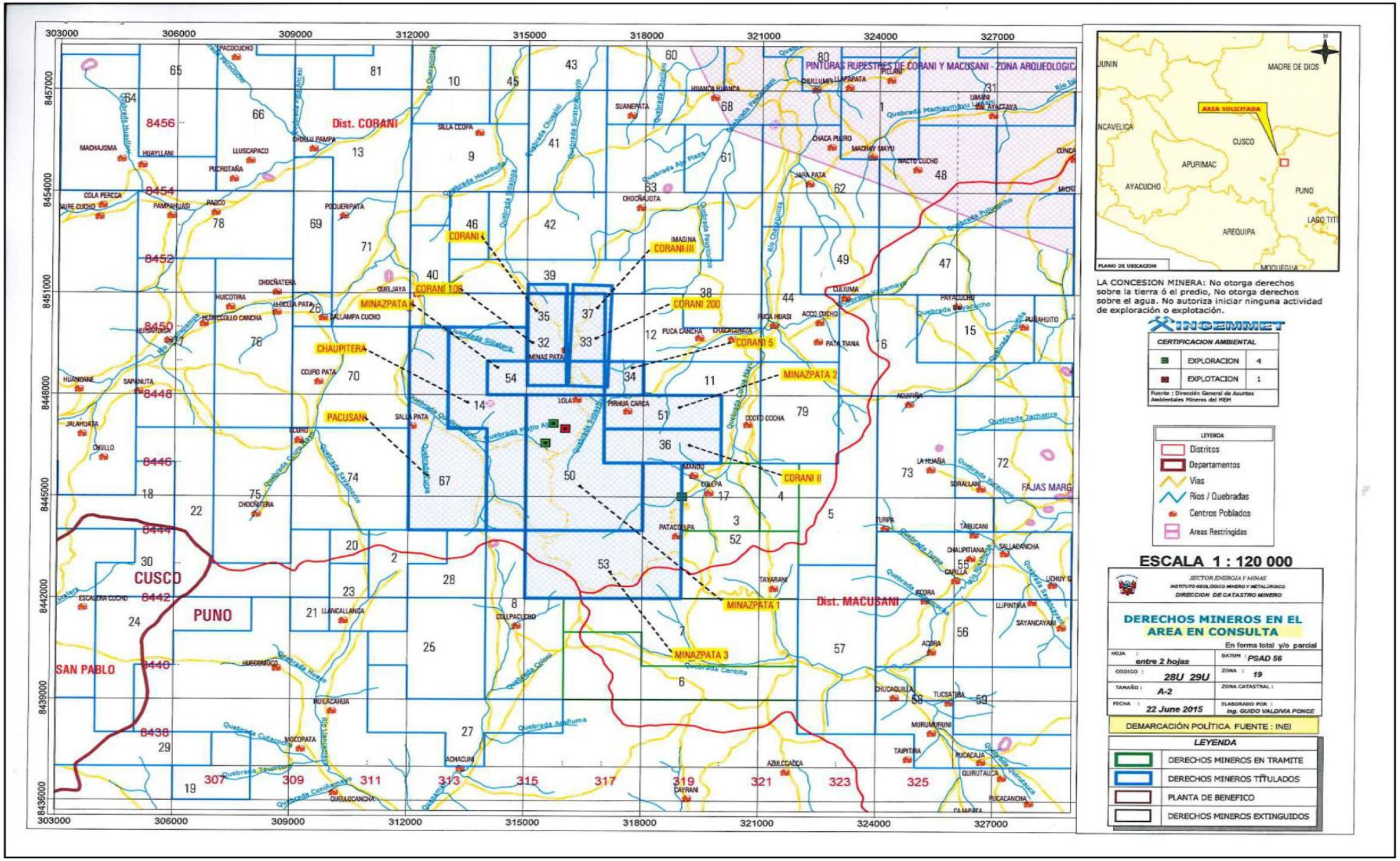
a. The Corani Project comprises the twelve (12) metallic mineral concessions (collectively the "Corani Project").

b. The Corani Project is located in the district of Corani, province of Carabaya, department of Puno, in Peru, and covers an aggregate available extent of 5,180.1213 hectares.

c. The location of the Corani Project is fixed, for all legal purposes, by the UTM coordinates (Datum PSAD 56) for each of their vertices shown on documents recorded in the Public Registry.

Figure 4-1 shows the location of the project within Peru. Figure 4-2 shows a map of the Corani mineral concessions within the area.

Figure 4-2: Map of Corani Mineral Concessions



4.2.4 Maintenance Obligations

According to the Estudio Grau (2015):

The twelve (12) mineral concessions comprising the Corani Project are subject to compliance with payment of annual license fees in the amount of US$3.00 per hectare (ha) ("License Fees").

The mineral concessions comprising the Corani Project are also subject to compliance with either of the following alternative obligations: minimum required levels of annual production of at least US$100 per ha in gross sales ("Minimum Production"); or payment of an additional amount referred as Penalty of US$6.00 per ha for the 7th through 11th year following the granting of the concession, and of US$20.00 per ha thereafter; or exploration expenditures of 10 times the Penalty. Compliance with one of these three maintenance obligations, together with timely payment of License Fees, is required to them in good standing. Failure to comply with License Fee payments or Penalty payments for two consecutive years causes the forfeiture of the mineral concessions.

In the year 2018, the twelve (12) mineral concessions comprising the Corani Project shall be subject to the obligations of Minimum Production, Penalties and exploration expenditures in accordance with the maintenance regime in force as of October 2008 whereby:

i. The minimum production will be equivalent to one Tax Unit per year (approximately U.S. $1,333.00) per ha granted for metallic minerals, and 10% of one Tax Unit per year per ha granted for non-metallic minerals ("Minimum Production").

ii. Failure to attain Minimum Production will trigger the obligation to pay a penalty equivalent to 10% of the Minimum Production per year per ha, until the year in which the Minimum Production is attained.

iii. Year 2028 shall be the maximum deadline for the mineral concessions comprising the Corani Project to attain Minimum Production. Failure to do so will result in the forfeiture of these mineral concessions.

The twelve (12) mineral concessions comprising the Corani Project are part of an Administrative Economic Unit under the name of Corani ("UEA Corani") duly approved by Geologic Mining and Metallurgic Institute - INGEMMET as the competent governmental agency. Bear Creek is authorized to comply with its maintenance and reporting obligations to the Peruvian State, applicable to the Corani Project through the UEA Corani. This includes the right to report exploration expenditures incurred in one or more mineral concessions to the benefit of the UEA Corani as a whole.

Table 4-2 shows the projected annual amounts for each of the alternative maintenance obligations to keep the Corani Project in good standing from 2012 through 2016. BCM has completed all required payments for all mineral concessions in 2017.

Table 4-2: Corani Mineral Concessions Maintenance Obligations

Mineral Concession	Annual License Fees Years 2012-2016 (US$)	Alternative Annual Maintenance Obligations Years 2012-2016				
		Minimum Production in gross sales from 2012-2016 (US$)	Minimum Exploration Expenditures (US$)		Penalty Payment (US$)	
			2012-2013	2014-2016	2012-2014	2015-onwards
Corani I	900.00	30,000.00	18,000.00	60,000.00	1,800.00	6,000.00
Corani II	900.00	30,000.00	18,000.00	60,000.00	1,800.00	6,000.00
Corani III	900.02	30,000.74	18,000.44	60,001.48	1,800.04	6,000.15
Corani 100	15.00	500.00	300.00	1,000.00	30.00	100.00
Corani 200	65.92	2,197.30	1,318.38	4,394.60	131.84	439.46

Mineral Concession	Annual License Fees Years 2012-2016 (US$)	Alternative Annual Maintenance Obligations Years 2012-2016				
		Minimum Production in gross sales from 2012-2016 (US$)	Minimum Exploration Expenditures (US$)		Penalty Payment (US$)	
			2012-2013	2014-2016	2012-2014	2015-onwards
Corani 5	279.78	9,326.01	5,595.61	18,652.02	559.56	1,865.20
Minazpata 1	3,000.00	100,000.00	60,000.00	200,000.00	6,000.00	20,000.00
Minazpata 2	900.00	30,000.00	18,000.00	60,000.00	1,800.00	6,000.00
Minazpata 3	3,000.00	100,000.00	60,000.00	200,000.00	6,000.00	20,000.00
Minazpata 4	479.64	15,988.08	9,592.85	31,976.16	959.28	3,197.62
Chaupitera	2,400.00	80,000.00	48,000.00	160,000.00	4,800.00	16,000.00
Pacusani	2,700.00	90,000.00	54,000.00	180,000.00	5,400.00	18,000.00
Total	15,540.36	518,012.13	310,807.28	1,036,024.26	31,080.73	103,602.43

4.2.5 Legal Standing

According to Estudio Grau (2015),

(a) The twelve (12) mineral concessions comprising the Corani Project are valid and in good standing. Ownership and current status were verified in August 2017 through an electronic database search through INGEMMET.

(b) They were validly applied for and granted title to concession by the competent governmental authority.

(c) Each of the twelve (12) mineral concessions comprising the Corani Project is designated metallic as a mineral concession and allows its titleholder or lessee the exclusive right to explore and exploit all metallic minerals located within their internal boundaries. These mineral concessions are separate from the surface right.

(d) The mineral concessions comprising the Corani Project have been granted to the titleholders for an indefinite period of time, provided that maintenance obligations, including license fee payments, minimum production, investment and/or payment of applicable penalties are attained when due. The year 2028 is the current legal absolute limit as to when production needs to occur with respect to the mineral concession comprising the Corani Project; failure to do so will cause their termination or expiry. The mineral concessions comprising the Corani Project will therefore remain valid through the maximum legal deadline to be put into production as long as the titleholder or lessee continues complying with annual license fee payments, qualified investments, and/or applicable penalties.

(e) Exercise of the rights derived from the twelve (12) mineral concessions comprising the Corani Project, including the right to explore, develop and further exploit, on an exclusive basis only the designated minerals within the internal boundaries of the mineral concession, is subject to the awarding of the required permits, authorizations and approvals, including relevant surface lands.

4.3 Surface Rights

BCM controls the surface rights that cover the entire project area including the open pit, waste dump, process plant, water ponds, camp, and ancillary facilities required for operation. Surface rights total 2,424 hectares.

4.4 Environmental Liabilities

Historical mining activities have been carried out in the vicinity of the proposed mine and associated facilities. The history of the Project site including ownership and any known mineral exploration and production are described in Section 6 of this report.

In accordance with Peruvian Law 28271, generators of environmental liabilities are responsible for remediation activities. Therefore, if historical environmental liabilities are defined, responsibility for these lies with the generator; however, BCM can assume responsibility for them in order to expedite the development of the site.

In December 2010, Walsh Environmental (Walsh) undertook an environmental-liabilities study in order to declare to the Ministry of Energy and Mines (MINEM) the existence of liabilities left from previous mining activities. These previous mining activities have left excavations, stopes, test pits, and mine portals on the site. During the study site visit, the location of each liability (either previously known or discovered during the study) was inventoried and registered with the MINEM. A total of 141 liabilities were recorded; however, it is possible there are others. A number of the environmental liabilities are located within the boundaries of proposed project components; therefore, in order for ground-breaking and or development activities to occur, BCM will need to assume responsibility for these.

Environmental liabilities associated with development of the property (past and future) are managed through an Environmental Closure Plan or Plans. Environmental Liabilities Closure Plan or Plans were approved by the Peruvian Government in April 2015 and must be reviewed within three years following the Peruvian legislation.

4.5 Permitting

BCM obtained the permits required for the previous field exploration activities and have identified the permits required for the construction, exploitation, and closure phases. An outline of the national, territorial, and municipal legislation, and the associated approvals and permits which apply to the Project, has been compiled and is provided in the Permitting Handbook for the Project (Vector, 2009d). It is considered that there has not been any material change in the permit requirements, notwithstanding some changes to the Project description. Table 4-3 presents a summary of the permits required.

Table 4-3: Summary of Permit Requirements by Phase

	Construction	Exploitation	Closure
ESIA *(modifications through the life of the mine may apply)*	X		
Certificate of Mining Operations *(for explosives use purposes)*	X		
Positive technical opinion of DIGESA *(for the construction of landfills outside the area of mining concessions)*	X		
Closure Plan *(modifications through the life of the mine may apply)*	X		
Certificate of Non-Existence of Archaeological Remains – CIRA	X		
Surface water use license	X		
Groundwater use license	X		
Sanitary authorization for wastewater treatment system and discharge	X		
Sanitary authorization for drinking water treatment system	X		
Fuel Direct Consumer's favorable technical report	X		
Registration as a direct consumer of liquid fuels - fixed or mobile facilities	X		
Authorization for eventual explosives use	X		
Explosives shack operation license	X		
License for explosives handlers	X		
Authorization for explosives transportation	X		
Identification code for users of Restricted Chemicals	X		

	Construction	Exploitation	Closure
Verifying deed for the purchase and transportation of IQPFs used by companies	X		
Authorization for opening Special Registry of IQPFs	X		
Incorporation in the Unique Registry for IQPFs	X		
Monthly reports of IQPF Special Registry	X		
Annual Opinion for making, marketing, and warehousing explosives of civilian use and related goods	X		
Definitive Concession for energy transmission line	X		
Individual license for radioactive facilities' handling	X		
Installation license for the operation of fixed nuclear measuring equipment	X		
Transportation Guide of hazardous materials and wastes	X		
Insurance for transportation of hazardous materials and wastes	X		
Certification of transportation personnel	X		
Registry for ground transportation	X		
Special driver's license	X		
Beneficiation Concession		X	
Authorization to start operation		X	
Posting Financial Assurance		X	
Final Closure Plan *(2 years before final closure)*			X
Final Closure Certificate *("Exit ticket")*			X

4.6 Water Supply

The updated water balance and water model prepared will be used to prepare a technical document required to obtain the authorization and permit from the Water National Authority (ANA), designated approving authority. This is the regular procedure in Peru.

4.7 Environmental and Permitting

The main environmental approval required in order to begin mining activities is an Environmental and Social Impact Assessment (ESIA). In 2013, the Ministry of Energy and Mines approved the ESIA based on the Feasibility Study prepared in 2011. The Closure Plan was approved in April, 2015.

The design and operating improvements incorporated in this Technical Report are expected to require only a modification of the existing approved ESIA, without the necessity for additional public hearings, as they are entirely located within the previously approved Project footprint. Furthermore, the environmental impact of the proposed Corani operation has been reduced as a result of the modifications described within this report, and the Company anticipates final permitting timelines will shorten and costs will be lower than previously anticipated.

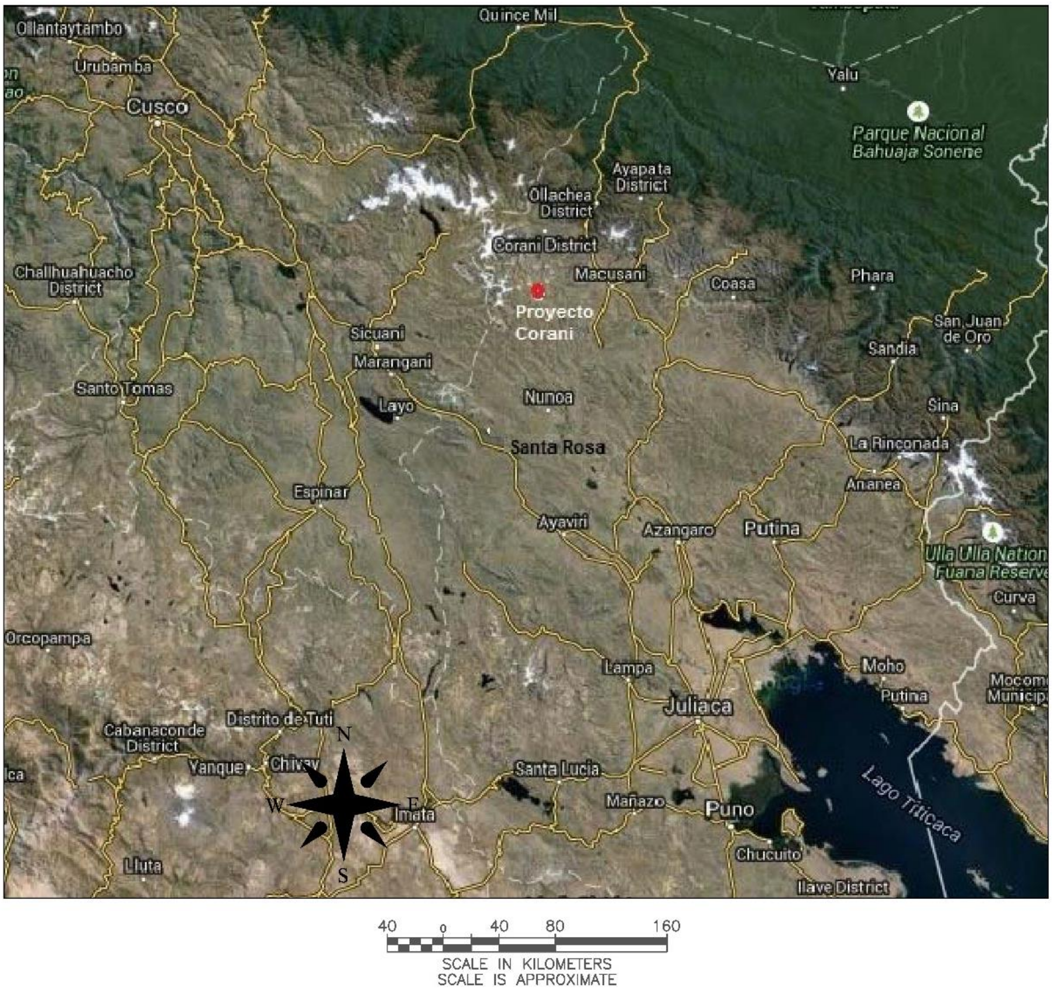
Bear Creek's plans for the Corani Project are to focus on preparing for development of the Project starting with the preparation and submission of an amended ESIA in the third quarter of 2017.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

Existing access to the Project site is primarily by road from the town of Macusani (located on the paved dual lane Interoceanic Highway), which is more readily accessible from the town of Juliaca, also serviced by commercial airlines from Lima. This route typically takes 4.5 to 5 hours and is also indicated on Figure 5-1. From Juliaca, the route generally aligns north towards the city of Azángaro on the paved Interoceanic Highway. The Interoceanic highway extends approximately 180 km between Azángaro and Macusani. At Macusani, the route extends west and northwest for approximately 60 km to the mine site on improved gravel roads.

Figure 5-1: Map of Existing Access to the Project



There are other access routes to the site from Cusco, taking approximately 6 hours by vehicle on increasingly primitive roads approaching the site. The access route is shown in Figure 5-1 and passes through Sicuani and the town of Santa Rosa, which is located approximately 208 km southeast of Cusco on a good paved road. From there, the route extends approximately 33 km northeast on improved gravel roads to the village of Nuñoa, and continues northeast for around 27 km on a less improved gravel road to the small village of Huaycho. From Huaycho, the access route continues north on an unimproved gravel road for approximately 70 km and ascends a mountain pass to the Project site. The City of Cusco is also serviced by commercial airlines from Lima.

5.2 Local Resources and Existing Infrastructure

5.2.1 General

The nearest town of significant size and with significant infrastructure is Macusani, which is the capital of the Province of Carabaya in the Region of Puno. Macusani is approximately 30 km east of the Project in a direct line. The access road from Huiquisa Bridge to the Permanent Camp will be improved. The length of the proposed Mine Access Road connecting the process Plant to Macusani is anticipated to be approximately 64 km.

Macusani has a total area of approximately 1,030 km^2 (no data was found specific to the urban area), and its economy is based mainly on agriculture and transportation of agricultural products.

Infrastructure in the town of Macusani includes a national highway - the Interoceanic Highway - currently complete from the Peruvian Port of Matarani to the town of Macusani. Other paved and unpaved roads, trails and footpaths allow access to most areas of the municipality.

The Project site is located in the district of Corani, also in the Province of Carabaya. The closely orientated campesino communities of Chacaconiza and Quelcaya, which have a joint population of approximately 200 families (80 and 120 families, respectively), will be directly impacted by the mine development in terms of landholding, rights to water, employment, etc.

Chacaconiza and Quelcaya are communities that maintain a fragile, high altitude economy. Both communities are below the poverty line with few resources for economic and social development. The main economic activity in these communities is the raising of alpacas. Approximately 90% of their economy is dependent on this activity, which is augmented to a very marginal degree by trading and seasonal migration.

5.2.2 Available Labor Force

The community consultation undertaken with the Chacaconiza and Quelcaya communities to date has included a proposal for mining employment, generating widespread acceptance, mainly among younger community members, the teachers at local educational facilities, and community leaders. The current labor force is generally unskilled, mainly working on highway remediation and maintenance. A technical training program is planned to develop the skills of community members to fulfill employment requirements of the Project, which will include agreements with universities and institutes to improve the local population’s vocational skill levels. The training program will include a system of scholarships that will allow the most successful students to occupy positions of greater responsibility on the Project. The training program has been designed to be conducted over a 5-year period. After this, BCM will continue to support the training program.

Mining and services related training will be segmented by age group, to allow older people to be trained for simpler tasks, while younger people will have access to jobs that demand more knowledge and specialization, such as the operation and maintenance of heavy machinery, woodworking and electrical work.

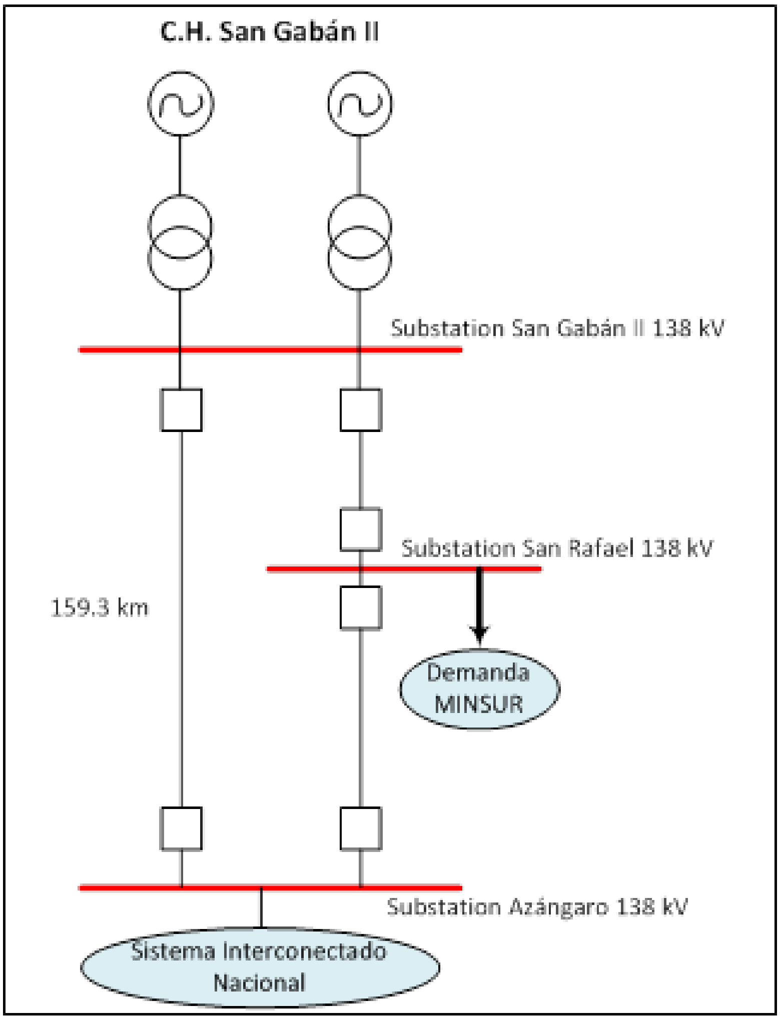
The Project’s requirement for labor will exceed the labor resources available in the Chacaconiza and Quelcaya communities. A ranking system will be developed with regard to geographical location of employment applicants, together with categorization and quantification of the labor force required.

5.2.3 Power

The National Interconnected Electric System (SEIN) is the source of power supply for the Project. The San Gabán II Hydroelectric Power Station is located on the San Gabán River, some 260 km north-west of the city of Puno and 100 km east of Cusco. The 138-kV power transmission line that connects the Hydroelectric Power Station of San Gabán II (CH San Gabán II) with the SEIN at the Azángaro Substation (SE Azángaro), passes through the neighboring areas of the Project, near the town of Macusani. Therefore, the Project's recommended access to power supply is from the SEIN, connected to this transmission line.

The main facilities that have a direct impact on the Project are described below, and are represented schematically in Figure 5-2.

Figure 5-2: Existing Facilities Schematic



San Gabán II Hydroelectric Power Station

This hydroelectric power station is owned by Empresa de Generación Eléctrica San Gabán, (EGESG), a state-owned company in charge of the operation of the plant's facilities since the end of its construction in the year 2000. The plant's characteristics are the following:

- Number of units : 2
- Power (each) : 54MW
- Energy (annual average) : 800GWh

138 kV transmission line CH San Gabán 2 – Substation Azángaro

The energy produced by the hydroelectric power station is delivered to the SEIN at the Substation Azángaro. The characteristics of this transmission line are the following:

- Number of circuits: 2
- Denomination of the circuits:
  - L-1010 (first circuit)
  - L-1013 and L-1009 (second circuit)
- Tension: 138 kV
- Length: 159.3 km
- Capacity by circuit: 120 MVA

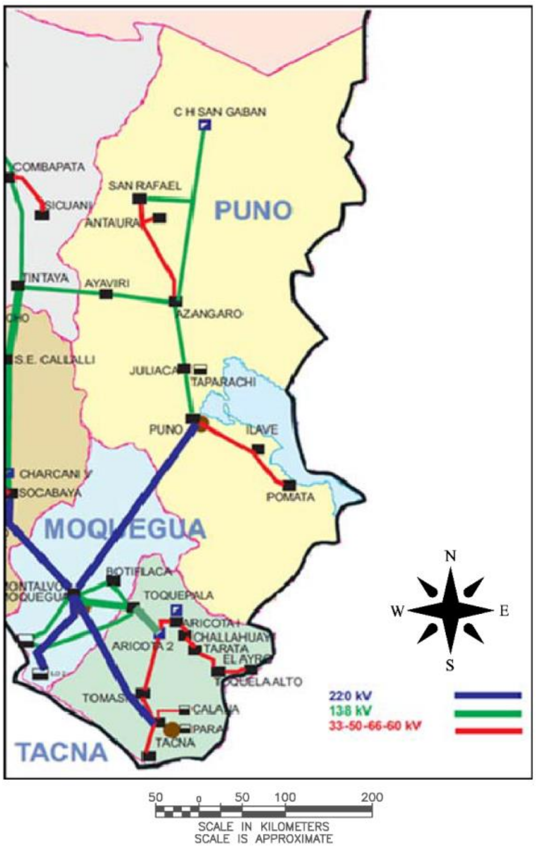
Azángaro Substation

The Azángaro substation is a state-owned substation, granted in concession to Red Eléctrica del Perú (REP). It is part of the southern transmission ring and has a three-winding transformer with a capacity of 12/12/5 MVA and voltage of 138/60/22.9 kV. The Substation San Rafael is also connected from this substation through a 60-kV transmission line, which is out of service at the time of this writing.

Southern Transmission Ring

The Peruvian Southern Transmission Ring interconnects the cities of Azángaro, Juliaca, Puno, Moquegua, Arequipa, Tintaya and Ayaviri. The Southern Transmission Ring is represented in Figure 5-3. At the Socabaya Substation of Arequipa city, the southern ring gets interconnected with the electrical system of the center-north, integrating the National Interconnected Electric System (SEIN). The lines between Puno, Azángaro, Tintaya and Arequipa are 138kV and the lines between Puno, Moquegua and Arequipa are 220 kV.

Figure 5-3: Southern Electric Ring Schematic



5.2.4 Water

Surface water in the region is typically taken for farming and livestock watering, as well as essential human needs, such as a drinking and bathing. Surface water and water from springs and marshlands is collected in catchments in the communities of Chacaconiza and Quelcaya and is distributed to basic water supply systems which conduct water to distribution points located in public squares of each respective community.

Nearby the Project site, hydraulic works related to irrigation or water storage were not observed. It was noted that local inhabitants have constructed handmade canals to irrigate pastoral areas, although the structures are considered temporary and many have been abandoned depending on irrigation requirements.

5.3 Climate

The Corani Project metrological station is located in the vicinity of the proposed plant site. Six years of data are available since the station was commissioned in December 2008.

The climate at the Project site is characterized by an estimated average annual precipitation of 635 mm, with the highest values recorded between October and April (89% of the annual precipitation). The annual average evaporation was determined to be on the order of 1,415 mm, with the highest monthly evaporation rates occurring in October (145 mm) and the lowest monthly evaporation occurring in April (87 mm).

The average annual temperature was 1.4°C. The maximum average monthly temperature was 4.0°C during the month of February, while the minimum average monthly temperature was 1.2°C in July. The lowest recorded temperature was -20.8°C in the month of March.

The average relative humidity is around 68%, with monthly averages ranging from a low of 47% in June to a high of 83% in February. The annual average wind speed is estimated to be 2.3 m/s, with monthly averages ranging from 1.7 m/s in February to 2.6 m/s in September. The wind direction is generally from the southeast.

Limited comparison of the site data may be drawn to other weather stations in the region. Several regional weather stations have relatively long data records. However, all of the available stations are a significant distance from the Project, and only general seasonal trends correlate with the Project meteorology station.

Firsthand observation of weather conditions during the 2011 drilling program indicates conditions vary significantly across the Corani Project site. Terrain and elevation have been observed to exert considerable influence on precipitation, temperature, and wind patterns.

Engineering designs should employ appropriate conservatism based on the limited site climate data available. The site climate will allow for year-round operations, with normal operating delays for conditions such as snow and fog. Freeze protection shall be required for all hydraulic works.

The operations can be completed year round at the site despite inclement weather during the months with high rainfall and/or snow.

5.4 Physiography and Vegetation

The Project site is located in the eastern Andes mountain range, between 4,600 and 5,200 m above sea level (masl). The area is characterized by mountainous terrain dominated by volcanic rock above which sits glacial gravel. The lithologic and climatic conditions have given rise to a series of cirques or bowl-shaped, steep-walled basins. During periods of rainfall, the valley floors collect precipitation allowing the generation of small wetlands (bofedales).

Apart from the vegetation associated with the wetlands mentioned above, areas of “puna” or alpine tussock grassland occupy the valleys and moderate to steep slopes. The areas above 4,700 masl mostly consist of steep mountainous slopes where erosion and climatic conditions largely prevent the development of soils or vegetation. These areas are scarcely vegetated with species specially evolved to withstand the harsh conditions. The naturally occurring acidic soils related to oxidation of sulfide bearing materials and the resulting ARD from exposed mineralized zones within the Project area has also prevented the development of vegetation where these conditions occur.

6 History

6.1 Prior Ownership and Production

Prior to the early 1950s, mineral exploration in the Corani district consisted of shallow prospect pits and adits in the northern portion of the current project area. These prospects are of unknown age and may date back to colonial Spanish time. Antimony prospects south and east of the property reportedly were active in the early 1900s, when there was limited antimony production (C.R. Petersen, 1967).

The first modern evaluation of silver-lead mineralization began with the location of mineral concessions in 1951 by Augusto Leon y Leon. In 1953, Fernando de Las Casas visited the site and prepared a geological report titled “The Negrominas – Corani District.” He mentioned that the rocks exposed in the area covered by the Negrominas Claims consist principally of a series of rhyolitic tuffs, breccias, and flows tilted to the Northeast. Also, he determined that two main types of ore-bearing structures are distinguished at Negrominas. Compañía Minera Korani was formed in 1956 to develop the silver-lead mineralization previously prospected. The mines were developed and operated from 1956 to at least until 1967; initially producing 80 tpd of ore. In 1967, Compañía Minera Korani was owned two-thirds by Compañía Minera Palca and one-third by M. Hochschild. Total historical production is uncertain, but is estimated at 100,000 tonnes of silver-lead-zinc ore. In early 1967, estimated mine production was reported at about 3,400 short tons per month, with grades of 7.0-9.0% lead, 2.3% zinc, and 8.0 to 11.0 oz/ton silver (C.R. Petersen, 1967).

Figure 6-1: Part of the Old Flotation Circuit

Historical maps of the underground workings show development on four levels (4820, 4843, 4860 and 4870 m levels for 50 meters vertically) that extend over an area of approximately 500 m in a general north-south direction (parallel to strike) by about 150 m in an east-west direction. It is not known when operations of Compañía Minera Korani ceased, but presumably they ceased in the late 1960s or early 1970s. This mining operation left behind many mine portals, waste piles, and mine tailing that continually produce ARD. Smaller portals are located approximately 2 km to the south of the historical mill and discharge ARD into the Collpa Mayo drainage.

The next exploration activity was by a private Peruvian company, Minsur. That exploration was reported to include 40 shallow drill holes in various locations, including a number of close proximity holes in the gold zone (located south of the current resource area). Although Minsur is an active mining company in Peru, attempts by BCM to secure copies of Minsur’s exploration data have been unsuccessful. None of Minsur’s exploration information is available or verifiable, although, reportedly, gold mineralization was encountered in some of Minsur’s drilling.

In late 2003 and early 2004, Rio Tinto Mining and Exploration began a surface exploration program for porphyry copper mineralization. During 2004, Rio Tinto conducted surface mapping, sampling, and ground magnetic surveys and developed access roads into the area. That initial work by Rio Tinto defined anomalous silver and lead mineralization to the south of the Korani mines and also defined a zone of anomalous gold mineralization in rock and soils.

The concession ownership by Compañía Minera Korani apparently lapsed during the 1970s. The ownership of Minsur also lapsed prior to Rio Tinto's exploration activities after 2000. Rio Tinto re-established some of the older concessions in their name beginning in 2003. BCM has added two concessions early in 2005 to create the current land position described in Section 4.

6.2 Historical Exploration and Estimates

There have been six previous mineral resource estimates for the Project and two estimates of mineral reserves, all of which have been published in previous technical reports. The six previous technical reports are summarized below:

1) March 31, 2006, National Instrument 43-101 Technical Report, Initial Resource Estimate for Corani Silver-Gold Exploration Project. SRK Consulting. Tucson, Arizona, United States.

2) October 4, 2006, Corani Project Mineral Resource Technical Report, Independent Mining Consultants, Inc, Tucson, Arizona, United States.

3) May 12, 2008, Technical Report, Corani Resource Estimate and PEA, Independent Mining Consultants, Inc. Tucson, Arizona, United States.

4) October 14, 2009, NI43-101 Technical Report, Prefeasibility Study Corani Project Puno Perú, Vector Perú S.A.C.

5) December 22, 2011, NI43-101 Technical Report, Feasibility Study. Corani Project. M3 Engineering & Technology Corporation, Tucson, Arizona, United States.

6) May 30, 2015, NI43-101 Technical Report, Optimized and Final Feasibility Study. Corani Project. M3 Engineering & Technology Corporation, Tucson, Arizona, United States.

The mineral resource and reserve estimates from each report are summarized in Table 6-1 through Table 6-6. The resource estimate from the 2015 Technical Report is identical to the estimate presented in this Technical Report; however, the reserve estimate differs and therefore the corresponding estimates of resources in additional to mineral reserves also differs.

Table 6-1: Mineral Resources - March 2006

Category	Kilotonnes	Silver g/t	Lead %	Zinc %
Measured	7,759	65.12	1.081	0.162
Indicated	20,123	43.61	0.678	0.251
Measured + Indicated	27,882	49.60	0.790	0.230
Inferred	87,627	72.91	1.032	0.578

Table 6-2: Mineral Resource - October 2007

Category	Kilotonnes	Silver g/t	Lead %	Zinc %	Silver Million ozs	Lead Million lbs	Zinc Million lbs
Main							
Measured	7,899	52.5	0.93	0.29	13.3	162.0	50.5
Indicated	44,196	40.7	0.70	0.39	57.8	682.0	380.0
Measured + Indicated	52,095	42.5	0.73	0.37	71.1	844.0	430.5
Inferred	11,898	49.7	0.64	0.26	19.0	167.9	68.2
Minas							
Measured	2,487	77.1	1.41	0.53	6.2	77.3	29.1
Indicated	39,405	52.2	1.03	0.40	66.1	894.8	347.5
Measured + Indicated	41,892	53.7	1.05	0.41	72.3	972.1	376.6
Inferred	20,713	47.3	0.74	0.30	31.5	337.9	137.0
Este							
Measured	14,558	82.7	1.07	0.76	38.7	343.4	243.9
Indicated	31,856	72.6	0.91	0.75	74.4	639.1	526.7
Measured + Indicated	46,414	75.8	0.96	0.75	113.2	982.5	770.6
Inferred	5,326	55.9	0.41	0.25	9.6	48.1	29.4
All Deposits							
Measured	24,944	72.6	1.06	0.59	58.2	582.7	323.5
Indicated	115,457	53.4	0.87	0.49	198.3	2,215.9	1,254.2
Measured + Indicated	140,401	56.9	0.90	0.51	256.5	2,798.6	1,577.7
Inferred	37,937	49.3	0.66	0.28	60.1	553.9	234.6

Source: (Published by Independent Mining Consultants, Inc. Based on 16 g/t Silver Cut-off Grade Contained Within an Approximate Open Pit)

Table 6-3: Historic Mineral Resource - May 2008

Category	Kilotonnes	Silver g/t	Lead %	Zinc %	Silver Million ozs	Lead Million lbs	Zinc Million lbs
Main							
Measured	10,025	42.3	0.80	0.37	13.6	176.8	81.8
Indicated	64,250	30.0	0.57	0.43	62.0	807.4	609.1
Measured + Indicated	74,275	31.7	0.60	0.42	75.6	984.2	690.9
Inferred	11,928	33.1	0.57	0.36	12.7	149.9	94.7
Minas							
Measured	6,168	53.4	1.05	0.44	10.6	142.8	59.8
Indicated	106,970	38.2	0.75	0.38	131.4	1,768.7	896.1
Measured + Indicated	113,138	39.0	0.77	0.38	142.0	1,911.5	955.9
Inferred	19,698	32.5	0.54	0.39	20.6	234.5	169.4
Este							
Measured	20,523	63.3	0.91	0.69	41.8	411.7	312.2
Indicated	40,485	52.0	0.75	0.57	67.7	669.4	508.7
Measured + Indicated	61,008	55.8	0.80	0.61	109.5	1,081.1	820.9
Inferred	1,526	30.4	0.41	0.21	1.5	13.8	7.1
All Deposits							
Measured	36,716	55.9	0.90	0.56	66.0	731.3	453.8
Indicated	211,705	38.4	0.70	0.43	261.1	3,245.5	2,013.9
Measured + Indicated	248,421	40.9	0.73	0.45	327.1	3,976.8	2,467.7
Inferred	33,152	32.6	0.54	0.37	34.8	398.2	271.2

Source: (Published by Independent Mining Consultants, Inc. Based on $9.35/t NSR Cut-off Grade Contained Within an Approximate Open Pit)

Table 6-4: Mineral Reserve and Resource - August 2009

Mineral Reserves, $9.10 NSR Cut-off					Contained Metal			Equivalent Oz	
Category	Ktonnes	Silver g/t	Lead %	Zinc %	Silver Million ozs	Lead Million lbs	Zinc Million lbs	Silver Million ozs	Silver g/t
Proven	27,957	70.2	1.08	0.59	63.1	665.7	363.6	115.0	127.9
Probable	111,666	54.3	0.90	0.43	194.9	2,215.6	1,058.6	360.3	100.4
Proven + Probable	139,623	57.5	0.94	0.46	258.0	2,881.3	1,422.2	475.3	105.9

Mineral Resources in Addition to Reserves, $7.85 NSR Cut-off					Contained Metal			Equivalent Oz	
Category	Ktonnes	Silver g/t	Lead %	Zinc %	Silver Million ozs	Lead Million lbs	Zinc Million lbs	Silver Million ozs	Silver g/t
Measured	10,791	16.7	0.43	0.45	5.8	102.3	107.1	16.2	46.8
Indicated	99,626	20.6	0.45	0.39	66.0	988.4	856.6	158.2	49.4
Measured + Indicated	110,417	20.2	0.45	0.40	71.8	1,090.7	963.7	174.4	49.1
Inferred	34,215	32.4	0.54	0.34	35.6	407.3	256.5	69.0	62.7

Note: for this reserve resource statement silver equivalency calculation represents the contained equivalent silver ounces sent to concentrate and is based on the resource metal prices assumptions of $13.00/oz Ag, 0.70/lb Pb and 0.65/lb Zn and recoveries to concentrate of 74.5% for silver and 71.7% for lead and 71.3% for zinc. The calculation does not take into account the net smelter return

Table 6-5: Mineral Reserve and Resource - October 2011

Mineral Reserves, $10.54/tonne NSR Cut-off					Contained Metal		
Category	Ktonnes	Silver g/t	Lead %	Zinc %	Silver Million ozs	Lead Million lbs	Zinc Million lbs
Proven	30,083	66.60	1.041	0.603	64.4	690.4	399.9
Probable	126,047	50.73	0.872	0.467	205.6	2,422.6	1,297.7
Proven + Probable	156,130	53.79	0.904	0.493	270.0	3,113.0	1,697.6

Mineral Resources in Addition to Reserves, $9.20/tonne NSR Cut-off					Contained Metal		
Category	Ktonnes	Silver g/t	Lead %	Zinc %	Silver Million ozs	Lead Million lbs	Zinc Million lbs
Measured	10,878	17.50	0.380	0.330	6.1	91.1	79.1
Indicated	123,583	20.80	0.380	0.290	82.6	1,035.3	790.1
Measured + Indicated	134,461	20.5	0.380	0.290	88.7	1,126.4	869.2
Inferred	49,793	30.00	0.464	0.278	48.0	509.4	305.2

Metal Prices: for Mineral Reserve - $18.00/oz silver, $0.85/lb lead, $0.85/lb zinc
for Mineral Resource - $30.00/oz silver, $1.00/lb lead, $1.00/lb zinc

Table 6-6: Mineral Reserve and Resource – May 2015

Mineral Reserves, Variable $23.00-11.00 NSR Cut-off							
Category	Ktonnes	Silver g/t	Lead %	Zinc %	Silver Million oz	Lead million lb	Zinc million lb
Proven	19,855	69.1	1.09	0.72	44.1	478.7	313.4
Probable	117,843	48.6	0.88	0.57	184.3	2289.2	1470.7
Proven & Probable	137,698	51.6	0.91	0.59	228	2,768	1,784
Mineral Resources in Addition to Reserves, $11.00 NSR cut-off, 15 g/tonne Ag Cut-off (oxide)							
Category	Ktonnes	Silver g/t	Lead %	Zinc %	Silver million oz	Lead million lb	Zinc million lb
Measured	14,360	32.01	0.34	0.19	14.8	108.4	61.6
Indicated	83,749	25.37	0.37	0.28	68.3	682.2	512.8
Measured + Indicated	98,109	26.34	0.37	0.27	83.1	790.6	574.4
Inferred	39,953	37.20	0.58	0.40	47.8	510.6	352.4

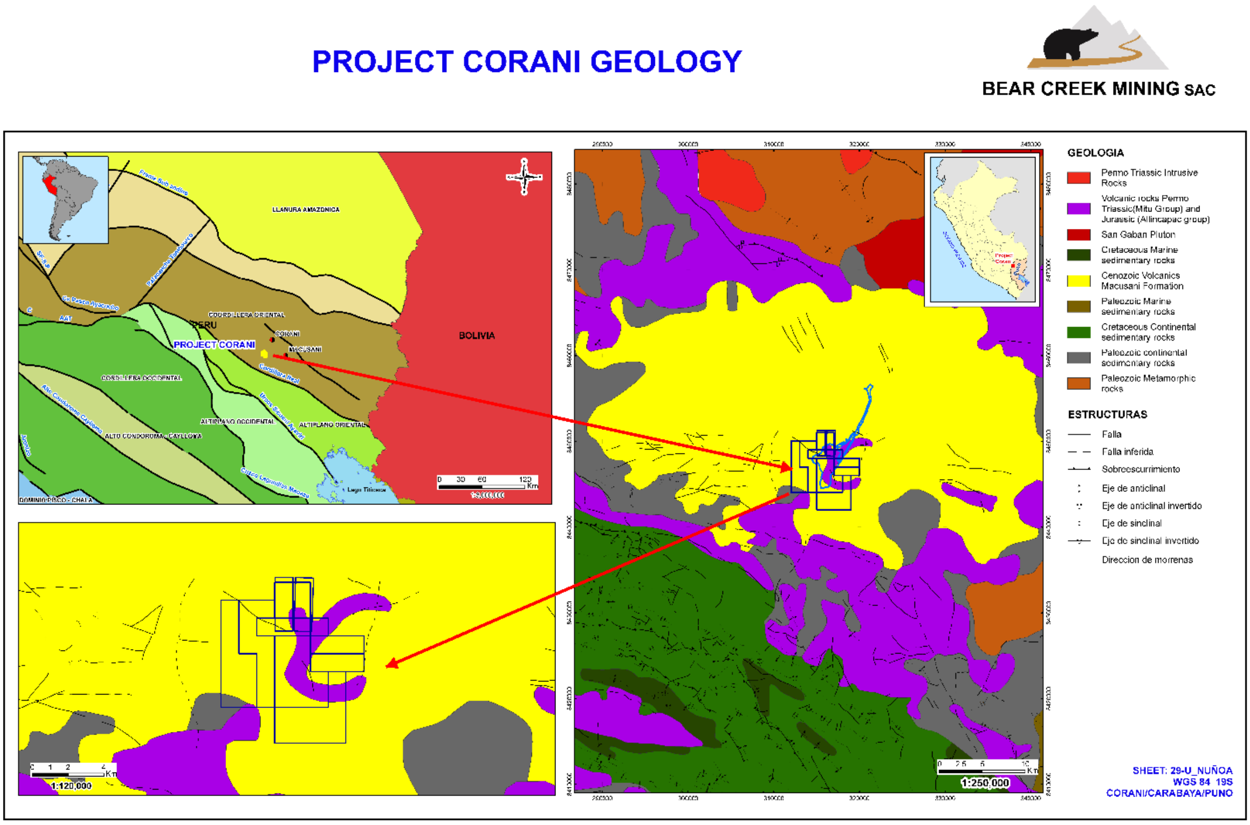
Notes: The Mineral Reserve is within the 20 $/oz designed pit and utilizes variable NSR cut-off values to maximize early cash flows. This is the tonnage processed in the economic model. The Mineral Resource is the tonnage contained within the 30$/oz silver, 1.425 $/lb lead, and 1.50 $/lb zinc prices Whittle pit using a 20 $/oz silver, 0.95 $/lb lead, and 1.00 $/lb zinc prices at a cut-off of 11 $/tonne NSR plus potentially leachable oxide at a 15g/t Ag cut-off ($4.80/tonne using 50% recovery in addition to ore already categorized within the Mineral Reserve.

7 Geology Setting and Mineralization

7.1 Regional Geology

The Corani Project area is located in the northern part of Puno Department, southern Peru, within the Cordillera Oriental of the Central Andes (Figure 7-1). The Cordillera Oriental in this region is represented by northwest-trending, glaciated peaks ranging in elevation from 5800 to 6400 m above sea level. Including its western foothills, the Cordillera Oriental forms a ~125 km-wide mountainous zone eroded into variably deformed Lower Paleozoic to Mesozoic metasediments (Laubacher, 1978) intruded by granitoid plutons of Triassic-to-Early Jurassic age (Kontak et al., 1990 and 1991).

Figure 7-1: Regional Geologic Map, Corani Project Vicinity



Throughout the region, the Lower Paleozoic is represented by the Ananea Group, a thick (up to 10,000 m) sequence of predominantly turbiditic sediments. These rocks were subsequently deformed during the early phase of the Hercynian Orogeny in Early Carboniferous time (340 Ma). The Upper Paleozoic is characterized by the accumulation of the Ambo Group, a thinner but lithologically more variable sequence of sedimentary rocks. The Ambo Group is comprised of sandstone, conglomerate, and minor carbonaceous layers of mostly continental derivation and was deposited unconformably upon the Lower Paleozoic strata in a post-Hercynian basin.

During the Lower Carboniferous, carbonates, shales, and sandstones of the Tarma Group were deposited in isolated basins; during the Upper Carboniferous, Copacabana Group carbonates were deposited over an extensive epicontinental area. The Mitu Group, comprised of continental redbed sandstone and conglomerate with volcanic intercalations, was deposited during the Permo-Triassic period, and the interval from the Triassic to the Upper Cretaceous is characterized by basin carbonate deposition and volcanism.

Magmatism occurred in several widely-separated episodes during the Mesozoic and Cenozoic (Clark et al., 1990). Intrusive activity was most active during Late Cretaceous and Early Tertiary, and volcanism began to dominate after Middle Tertiary. Tertiary volcanic activity is largely represented by the Oligocene to lowermost Miocene Picotani Group and the Lower to Upper Miocene Quenemari Group. The Picotani Group is comprised of rhyodacites intercalated with K-rich basalts and ultrapotassic, lamproitic, and lamprophyric flows, whereas the Quenamari Group is dominated by rhyolites and two-mica syenogranites (Li, 2016).

7.2 Local and Property Geology

The following description of the geology, structure, alteration, and mineralization specific to the Corani Project is largely modified from, and in some cases is excerpted directly from, Society of Economic Geologists Special Publication No. 15, "*The Discovery History and Geology of Corani: A Significant New Ag-Pb-Zn Epithermal Deposit, Puno Department, Peru*," prepared by Swarthout et. al. (2010). GRE has reviewed this information and available, associated supporting documentation in detail and finds the discussion and interpretations presented herein to be reasonable and suitable for use in this report.

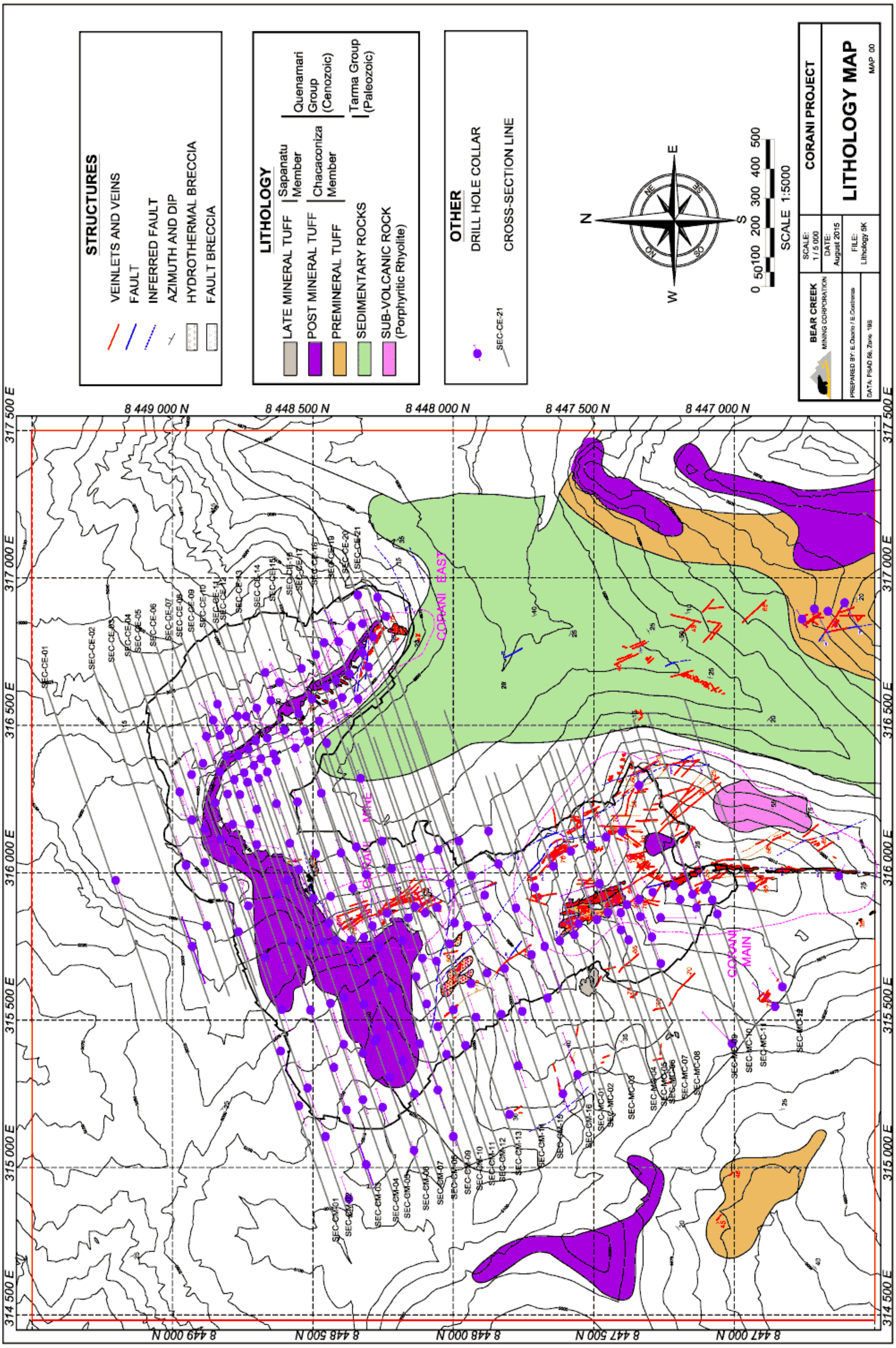
7.2.1 Lithology

The oldest strata encountered in the vicinity of the Project belong to the Ambo and Tarma Groups, which consist primarily of shales with minor quartzites, sandstones, and local carbonate lenses. Within the Project area, the sedimentary units are dominantly red to gray shales, which commonly contain syngenetic or diagenetic pyrite. These rocks are moderately folded and faulted, striking northwest with 10° to 50° dips east and west along the flanks of a modest anticlinal fold. Though outcrops of these rocks within the Project area are relatively few, resistant quartzite units often form ridges throughout the greater regional area, whereas weathered shales generally form slopes. The local lithology is illustrated on Figure 7-2.

The overlying Chacacuniza Member of the Quenemari Formation is the primary host of mineralization at the Corani Project. The Chacacuzina is the youngest member, ca. 23.94 ± 0.15 Ma (Ullrich, 2006), of the Quenemari and is comprised of a sequence of crystal-lithic and crystal-vitric-lithic tuffs. Within the Project area, these rocks strike northwest and dip subhorizontally to 40° northeast. Bedding ranges from well-bedded to massive, locally showing poorly developed columnar jointing. The tuffs are widely hydrothermally altered and pervasively argillized to low-temperature clays and are variably faulted, fractured, and brecciated. Individual units of the tuff sequence are pumiceous, and in general, the more lithic-rich units occur near the base, while the finest grained crystal tuffs are located within the upper part of the overall sequence. The upper portion of the tuff sequence includes a generally well-bedded andesite flow (or flows) of variable thickness. The andesite hosts mineralization in similar character to the surrounding tuffs, with no apparent constraints attributable to the change in lithology.

The Chacacuzina is unconformably overlain by the upper members (undifferentiated) of the Quenemari Formation, the Sapanuta and Yapamayo Members, ca. 10.375 ± 0.080 Ma (Ulrich, 2006). These rocks are nearly identical to the Chacacuzina in lithology, composed primarily of crystal lithic tuffs, but are notably different in both alteration and mineralization. They are generally flat-lying to shallowly dipping and are separated from the underlying tuff sequence by an angular unconformity with substantial topographic relief. The upper (Sapanuta/Yapamayo) tuff sequence is commonly collectively referred to as the "post-mineral" tuff, while the tuffs of the Chacacuzina are referred to as "pre-mineral." The post-mineral tuff occupies much of the volcanic section in the northern portion of the Project area and is largely unaltered and considered effectively barren.

Figure 7-2: Geologic Map of the Corani Project Area



7.2.2 Structure

The Corani Project area has been affected by brittle deformation at both local and regional scales. The geometry and kinematics of known structures suggest the occurrence of two tectonic events: a) an extensive tectonic event contemporaneous with mineralization within and to the south of Corani Main, locally expressed by listric faults that have generated a light tilting and rotation of the blocks; and b) a second extensional tectonic event resulting in post-mineralization structures such as those identified within and to the south of Corani Este (Ayala Prado, 2008).

The Corani deposit is hosted within a stacked sequence of listric normal faults striking dominantly north to north-northwest with moderate to shallow (50° to <10°) westerly dips. The hanging walls of the listric faults are extensively fractured and brecciated, providing the structural preparation for subsequent or syngenetic mineralization. The stacked listric faults are more prominent in the Corani Minas and Corani Main areas. The Corani Este area contains a single known listric fault with an extensively fractured and brecciated hanging wall. The contact with the underlying Paleozoic sediments corresponds locally to listric faults dipping shallowly to the west.

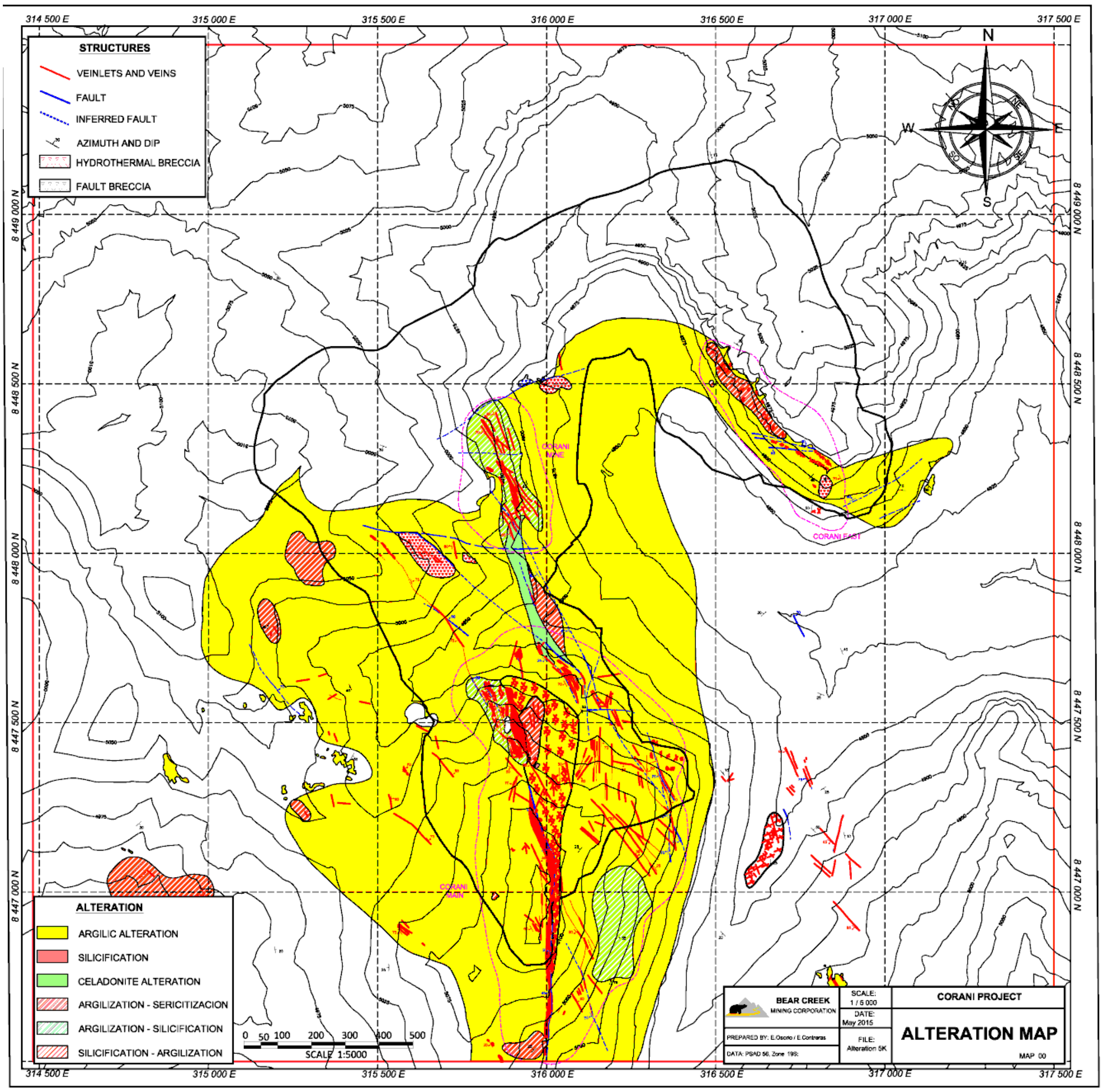
Desrochers (2005) described the structural geology of Corani as being dominated by a series of north- and north-northwest−trending faults and veins with Ag, Au, and Sb mineralization. Field observations indicate that the mineralized veins formed during normal movement along the faults, although the extension was accompanied by a minor amount of sinistral strike slip. The faulting moved the western, or hanging wall, block down and away from the main structure with a southwest-trending transport direction. Maximum extension took place at the intersection of the north- and north-northwest−striking structures, and in northwest-trending bends along the main north-south structure. These bends also coincide with steeper parts of the main north-south structure.

7.2.3 Alteration

Generally, illite-kaolinite alteration of the pre-mineral tuffs is present over a 5- by 2-km area, approximately occupying the entire window exposed beneath the post-mineral tuffs. Most common alteration phases associated with the mineralization are illite, kaolinite, and smectite/chlorite/celadonite. Distribution of dominant alteration types is shown in plant view on Figure 7-3.

Gangue minerals include massive to banded quartz, barite, chalcedony, and iron and manganese oxides. The mineralization and alteration assemblages are well documented in thin and polished sections (Gagliuffi Espinoza, 2006) and by Quantitative Evaluation of Minerals Scan (QEMSCAN) (Gunning, 2007). Each of the mineralized areas exhibits differences in alteration and gangue phases, which are described in further detail in Chapter 13 of this report.

Figure 7-3 Alteration Map



7.2.4 Mineralization

Mineralization at the Corani Project occurs in three distinct and separate zones: Corani Main, Corani Minas, and Corani Este, each differing slightly in character with regard to both alteration and mineral assemblages. In general, mineralization in outcrop throughout the Corani Project is associated with iron and manganese oxides, barite, and silica. Silicification is both pervasive and structurally controlled along veins. In drill core, the mineralization occurs in typical low to intermediate sulfidation Ag-Pb-Zn mineral assemblages.

The most abundant silver-bearing mineral is fine-grained argentian tetrahedrite or freibergite (Hazen Research, 2006; Gunning et al., 2007). Other minor silver minerals present include acanthite and the lead-silver sulfosalts, adorite and diaphorite. Other sulfide minerals include pyrite-marcasite, boulangerite, sphalerite, and galena. Boulangerite and galena do not appear to be significant hosts for the silver. Sphalerite, mainly high Fe type, overlaps the silver mineralization but can be more areally extensive, particularly at Corani Minas where sphalerite may extend 10 to 100 m beneath and lateral to the silver-bearing minerals. Lead also occurs as plumbogummite, a lead-aluminum phosphate. Lead mineral speciation is dependent on pH, and the plumbogummite is believed to be secondary in origin, forming as a result of the remobilization of lead in the presence of phosphate in a very acidic environment with abundant aluminum.

BCM geologists have defined nine specific mineral domains within the Corani deposit, based in part on metallurgical properties, as described below:

- CSC – Coarse-grained silica-sulfide-celadonite characterized by readily discernible sulfides (galena-sphalerite-chalcopyrite ± tetrahedrite) with celadonite in crystalline to locally opaline quartz with good Ag-Pb-Zn recoveries.
- CS - A subset of CSC that contains coarse galena-sphalerite-chalcopyrite ± tetrahedrite without green celadonite clay.
- TET – Ag-bearing tetrahedrite characterized by recognizable late-stage, coarse-grained tetrahedrite cutting earlier sulfides and displaying the highest Ag contents and best Ag recoveries by flotation or leach: typically with low Pb-Zn content.
- PM – Pyrite-marcasite ± quartz typical of low temperature early-stage mineralization with little polymetallic mineralization, mainly Zn.
- FBS – Fine-grained black silica-sulfides characterized by very fine-grained mineralogy deposited from quenched ore fluids with highly variable metal content and generally poor leach recoveries and good flotation recovery with some challenges in separation.
- QSB - Crystalline quartz-sulfide-barite interpreted as early fault fill or late-stage breccia fill.
- PG – Plumbogummite, identified as a pale-green, waxy, Pb-phosphate mineral that in metallurgical test results shows diminished lead flotation and difficulties in separation of base metals.
- FeO – Iron-oxide mineralization with locally elevated Ag and generally low Pb-Zn. This is a gradational zone with mixtures of FeO and FBS and the most strongly oxidized areas shows high Ag leach recovery results; and
- MnO – Manganese-oxide mineralization hosting mainly Ag with lesser Pb-Zn with very poor response to flotation and leach tests.

Mineralization is largely structurally controlled in each of the three areas along a general north-northwest strike. Figure 7-4 illustrates the distribution of silver mineralization based on thickness at an approximate cut-off grade of 15 g/t silver. Strike length of silver mineralization is roughly 2 km for Corani Main and Corani Minas combined, and 1.5 km for Corani Este.

The cross sections presented in Figure 7-5 illustrate the general thickness and dip of the mineralized zone(s).

All mineralization, with the exception of pyrite, dies out at depth, typically ending as much as 50 m above the contact with underlying Paleozoic basement. In drill core, the contact with the Paleozoic sedimentary rocks is locally sheared with gouge and slickensides. The sedimentary rocks are locally brecciated, but the breccias do not contain a hydrothermal matrix. The strata locally contain pyrite veins, with bleached (Fe-reducing) halos, and Fe carbonate veinlets (Nelson, 2006). Veining within the Paleozoic sedimentary rocks is completely barren of economic mineralization.

Corani Main

The primary mineralized vein breccia in the Corani Main area can be traced for 800 m and undulates with strikes and dips varying between S50°E, 55° W and S20°W, 40° W, with steepening dips to the north. Vein breccias locally attain widths of >10 m. To the north, strike and dip change, which indicates a plunging, dilatant structure that attains widths of 80 m, with near-vertical quartz veins surrounded by stockwork systems in adjacent wall rock. The change in strike suggests that an overall sinistral (left-lateral) strike-slip component affects the veining, causing the mineralization to blow-out, or widen and intensify, to the north (Nelson, 2006). Breccias and stockworks are characterized by chalcedonic, cockscomb, crystalline, hyaline, and amethystine quartz, with hematite-jarosite-goethite and barite stringers resulting in a highly banded texture. Manganese oxides are generally sparse but locally abundant. Pyrite, dark sphalerite, and freibergite are present (Petersen, 2005; Gagliuffi Espinoza, 2006). A post-mineral transverse fault, striking N70° E and showing dextral movement, separates the Corani Main and Corani Minas mineralized areas. The fault deforms the veins on either side; however,

Figure 7-4 Silver Mineralization Thickness at 15 g/t Cut-off Grade

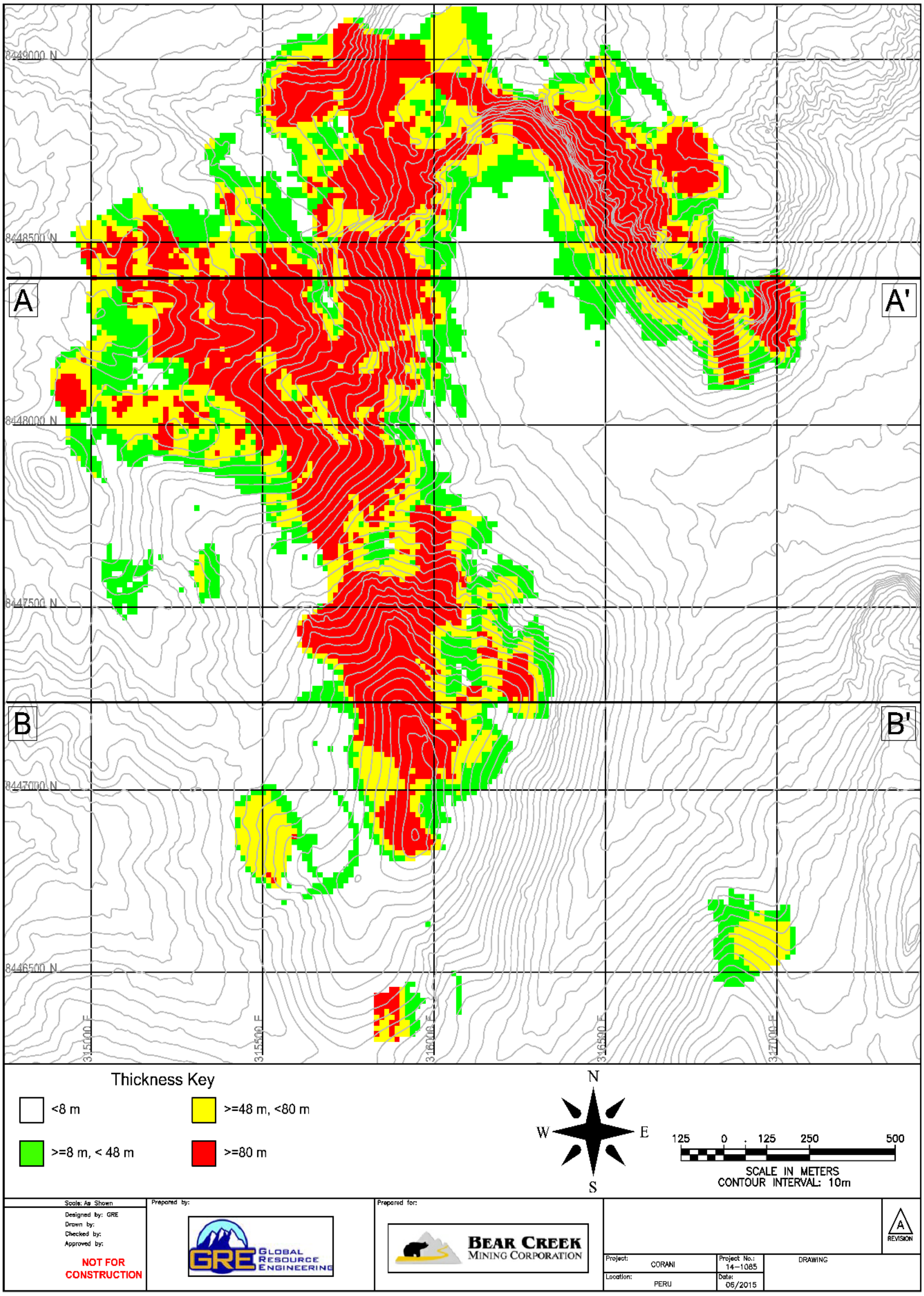
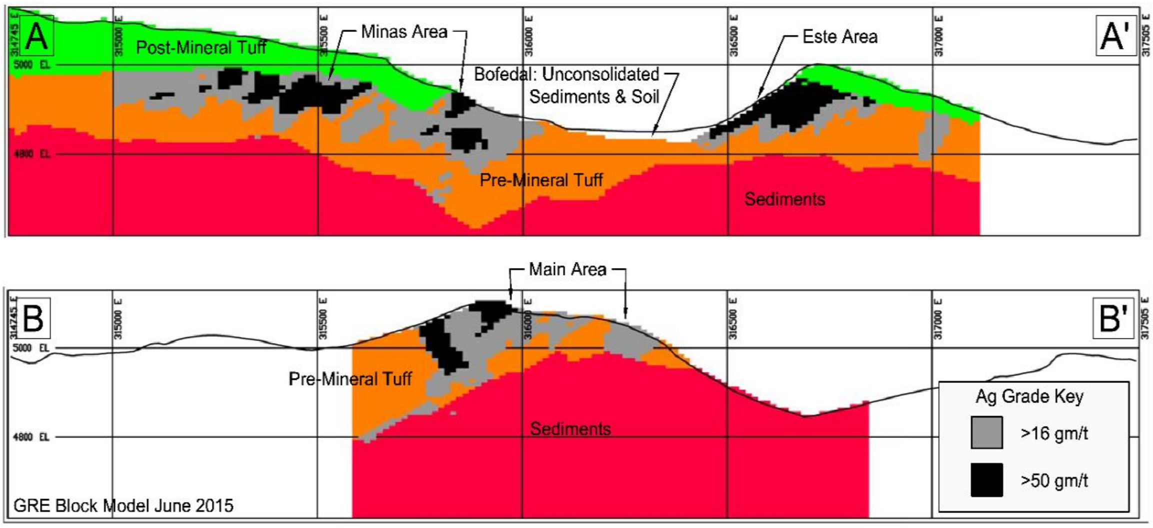
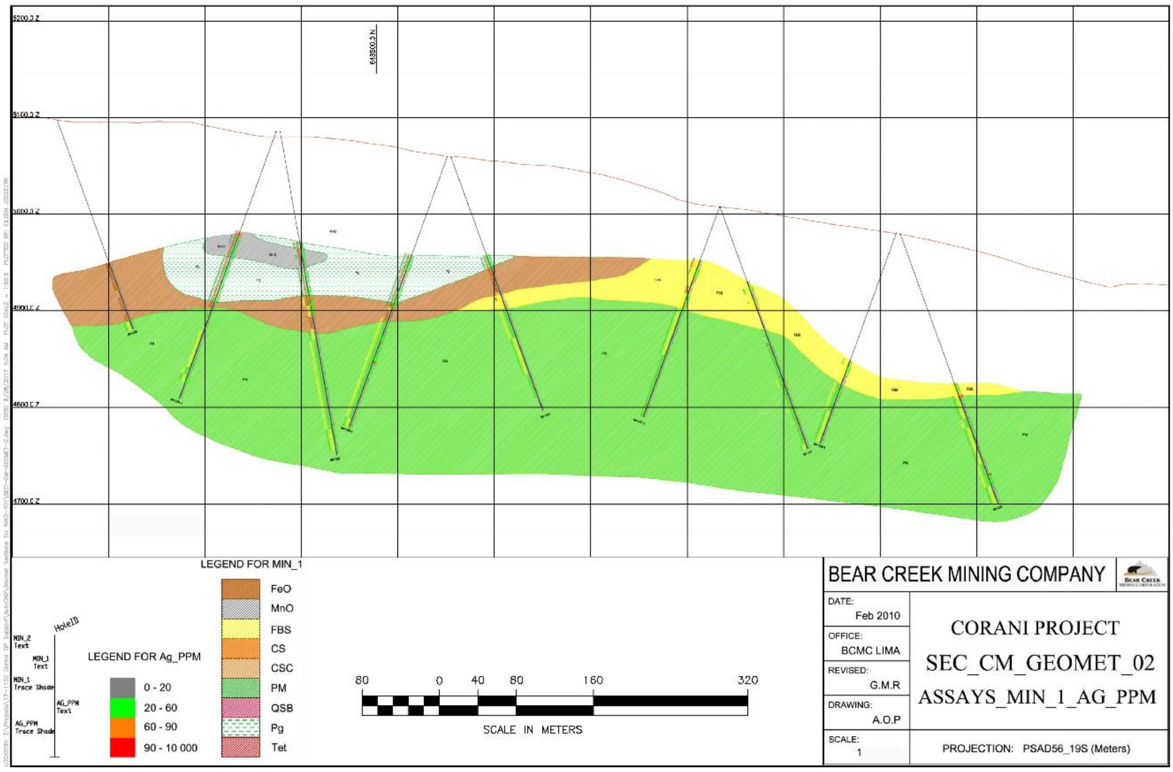


Figure 7-5 Silver Mineralization Distribution



post-mineral displacement appears minimal, despite the fault's appearance as a major lineament. Figure 7-6 shows a representative cross-section of Corani Main.

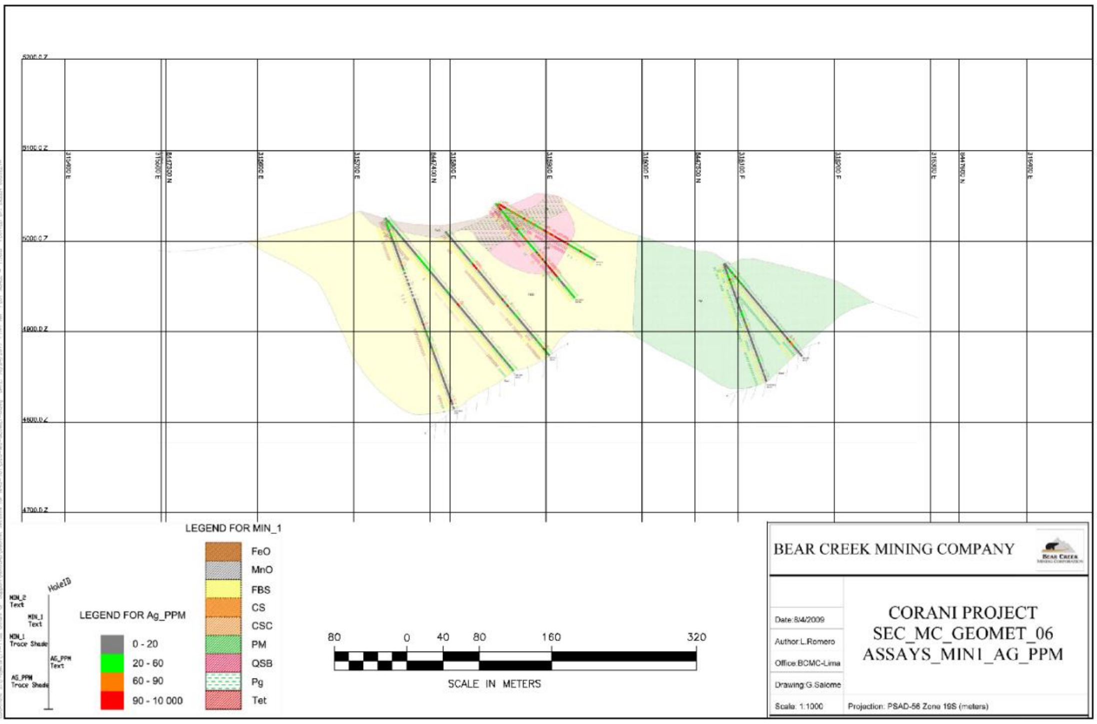
Figure 7-6: Corani Main Representative Cross-Section



Corani Minas

Corani Minas is structurally complex and characterized by a large area of small, crested ridges formed by breccias, silicification, and quartz vein ribs. Veins, 0.1 to 2.0 m wide, are composed of banded, chalcedonic, and hyaline quartz, barite, hematite, jarosite, goethite, pyrite, and proustite-pyrargerite. The veins generally strike north 20°−60° west with 50° to 80° dips to both the west and east (Desrochers, 2005; Nelson, 2006; Ayala Prado, 2008), although almost any dip angle can be observed (Nelson, 2006). Hanging-wall breccias are composed mainly of subrounded to angular clasts with void-filling barite crystals in a siliceous matrix, whereas stockwork veining is mainly within the footwall. According to Nelson (2006), orientations and textures, such as the absence of slickensides, indicate that veining was extensional and that block rotations tended to occur surrounding an east-west axis. Figure 7-7 shows a representative cross-section of Corani Minas.

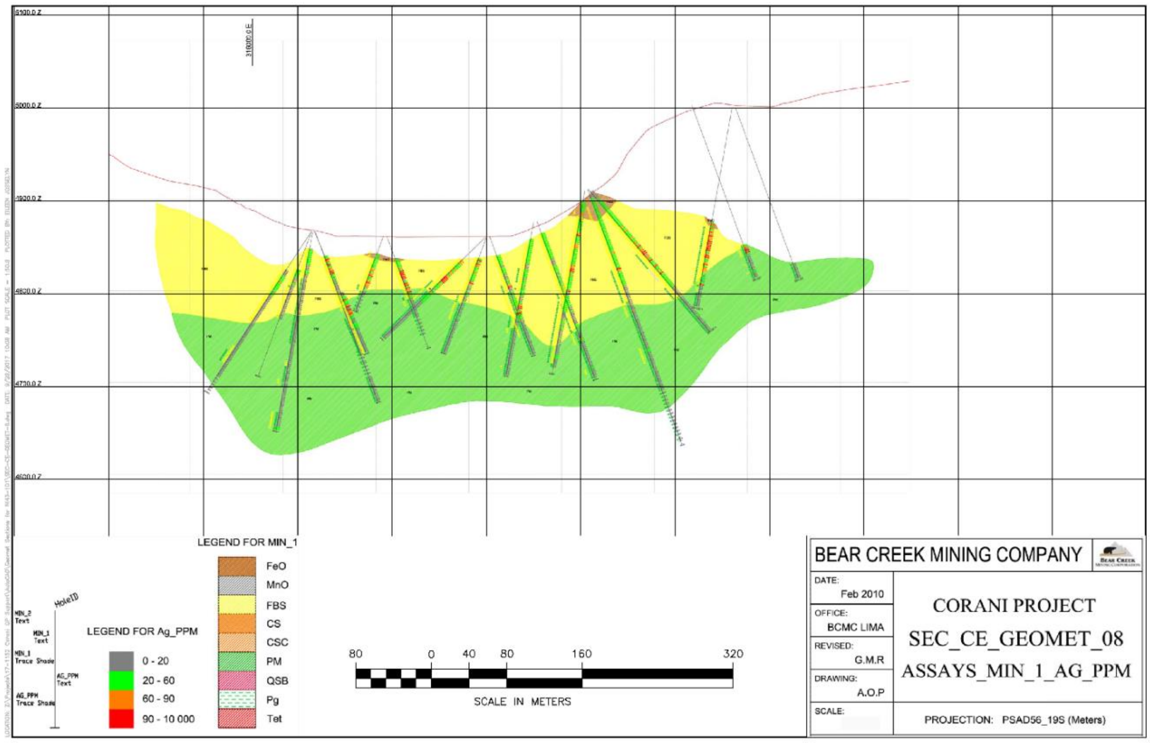
Figure 7-7: Corani Minas Representative Cross-Section



Corani Este

Corani Este is distinct in that mineralization is controlled by a single listric fault that does not crop out due to post-mineral tuff cover. Silicified breccias and stockwork veining are formed in the hanging wall of the main vein and crop out as silica-rich ribs along a north-south strike. Dips are difficult to determine in outcrop and drill core due to the structural complexity and hydrothermal alteration; however, conjugate vein sets, occasionally north-south striking and dipping both east and west, are observed. Importantly, a small breccia pipe occurs in Corani Este containing high-grade silver values (⩾300 g/t Ag). Mineralization within the pipe occurs in hydrothermal breccias with a dark-gray, sulfide matrix and dark-purple, jasperoid vein breccias. Figure 7-8 shows a representative cross-section of Corani Este.

Figure 7-8: Corani Este Representative Cross-Section



8 Deposit Types

The Corani deposit is best described as a low- to intermediate-sulfidation epithermal deposit with silver, lead, and zinc hosted in stock works, veins, and breccias. Epithermal deposits as originally defined are products of volcanism-related hydrothermal activity at shallow depths and low temperatures, with deposition normally within about 1 km of the surface and in the temperature range of 50 to 300°C. A few epithermal deposits can be related directly to deep-seated intrusive bodies, but this relationship is demonstrable only where especially deep erosion has occurred. Most ores of this type are in or near areas of Tertiary volcanism.

While epithermal, mineralization at Corani is likely both laterally and vertically distal to an intrusive source. Mineral textures grade from coarse crystalline quartz-pyrite-chalcopyrite in the southern portion of the Project area, to finer grained, pyrite-dominated sulfide minerals in the north, suggesting a south-to-north hydrothermal fluid flow. This spatial zonation suggests a rapidly cooled ore fluid typical of a distal setting surrounding a buried intrusion (Swarthout et al., 2010). As proposed by Swarthout et al. (2010), the multiphase nature of the mineralization and zonation at Corani may be related to multiple fluid exsolution events from an evolving porphyry type system that possibly underlies the southern part of the area. Alternatively, the mineralizing solutions may be related to shallow, subvolcanic dome emplacement.

Mineralization at Corani is principally located in a set of lístric faults dipping west (Desrochers, 2005; Nelson, 2006; Ayala Prado, 2008), showing dilation segments related to subvertical structures and breccias in the hanging wall. In the footwall, mineralization occurs as veinlets forming stockwork structures. Structural control of the mineralization is a product of extensional tectonics that developed north- to northwest-trending fractures and faults, and whose movements provided the structural preparation for the influx of mineralizing fluids. Geometric analysis (Ayala Prado, 2008) of the faults and fractures indicates that veins are placed in the steeper (sub-vertical) and more superficial portions of the greater system of normal faults. The listric faulting described by Desrochers (2005), Nelson (2006), and Ayala Prado (2008) provides a sound structural model that can and should be tested and refined with future drilling exploration and resource modelling.

9 Exploration

BCM began exploring the Corani Project in early 2005. In addition to drilling, which is discussed in detail in Chapter 10 of this report, exploration activities carried out by BCM include detailed geologic mapping, trenching, and geophysical surveying.

9.1 Geophysical Exploration

VDG del Perú S.A.C. (VDG) conducted a ground geophysical campaign at the Corani Project on behalf of BCM in the fall of 2005. A total of 44.20 line-km of induced polarization (IP) data was collected, along with 50.95 line-km of magnetic survey. The line layout was established by means of a Real-Time GPS prior to the geophysical surveys for a total of 51.65 line-km.

The geophysical surveys were aimed at assisting in geological mapping, including lithologies and key structures and at mapping mineralization and alteration associated with a low sulfidation gold-silver system. The objective of the IP/Res survey was to map the electrical response by means of high-resolution IP traverses across the favorable north-south corridor identified based on the results of both trench and drilling exploration. The chargeability is instrumental in defining disseminated sulfides associated with economic mineralization.

The field results of both methods were of good quality and were meaningful. The final chargeability and resistivity depth sections mapped systematically clear contrasts from line to line between the sub-surface and a nominal depth of 283 meters below surface. The chargeability outlined five (5) IP anomalies (Figure 9-1), two of which, IP1 and IP3, correspond to the Corani Main and Corani Este areas, respectively. Those anomalies accurately mapped the known mineralization and extended the size of both mineralized zones. A separate anomaly (IP4), located to the south and east of the existing resource area, exhibited the same type of geophysical response as the Corani Este anomaly. This anomaly extends for 1,600 meters by 400 meters, and remained open to the south. Another anomaly (IP5) extends to the south of Corani Main and appears to be the extension of it. This anomaly is a wide zone, some 600 by 600 meters, and is also open to the south. A reconnaissance survey completed in the southern part of the property along three lines spaced every 500 meters outlined encouraging chargeability anomalies that allowed extending the favorable prospective corridor to the south (Figure 9-2).

Figure 9-1 Geophysical Interpretation Map

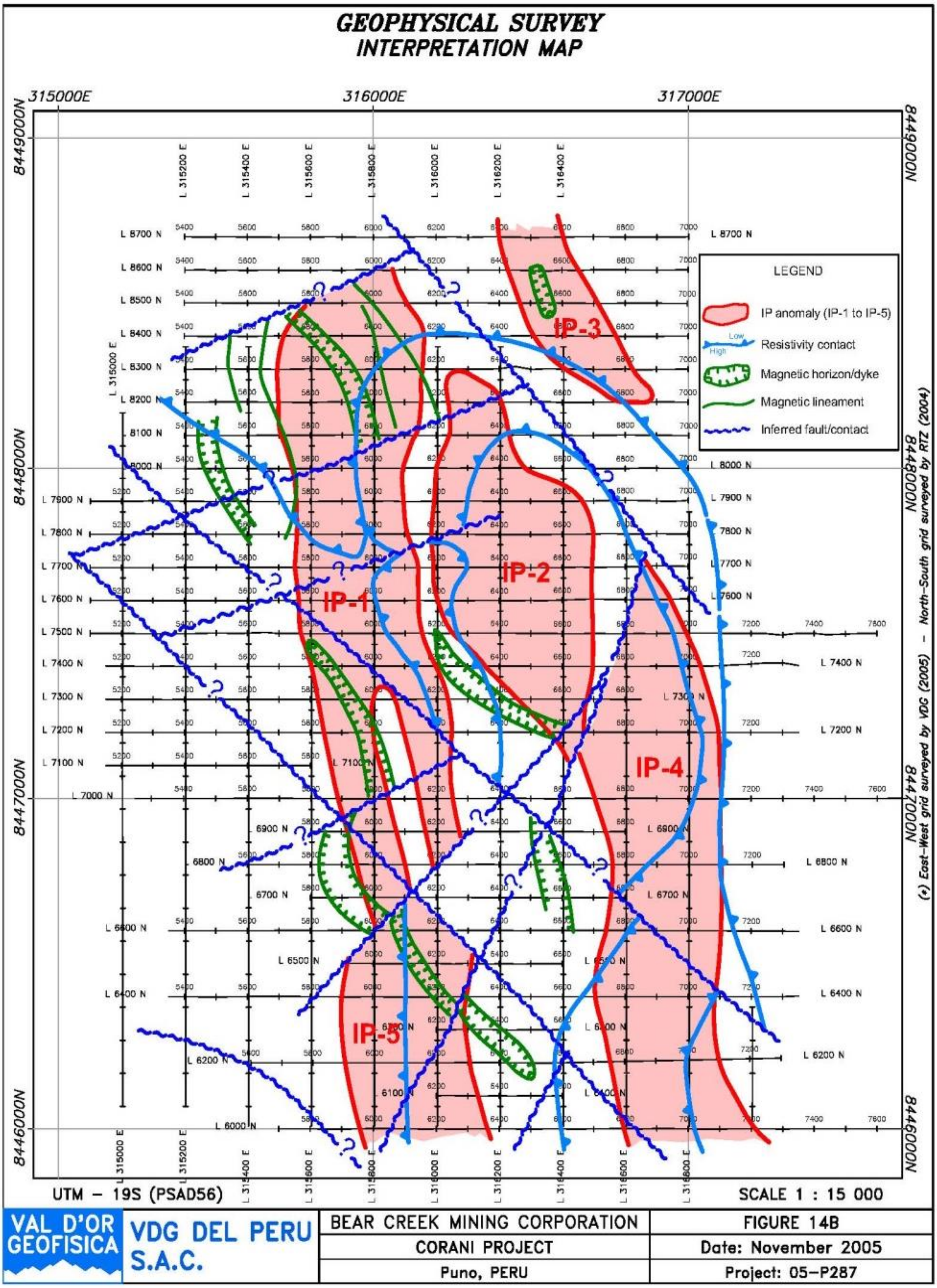
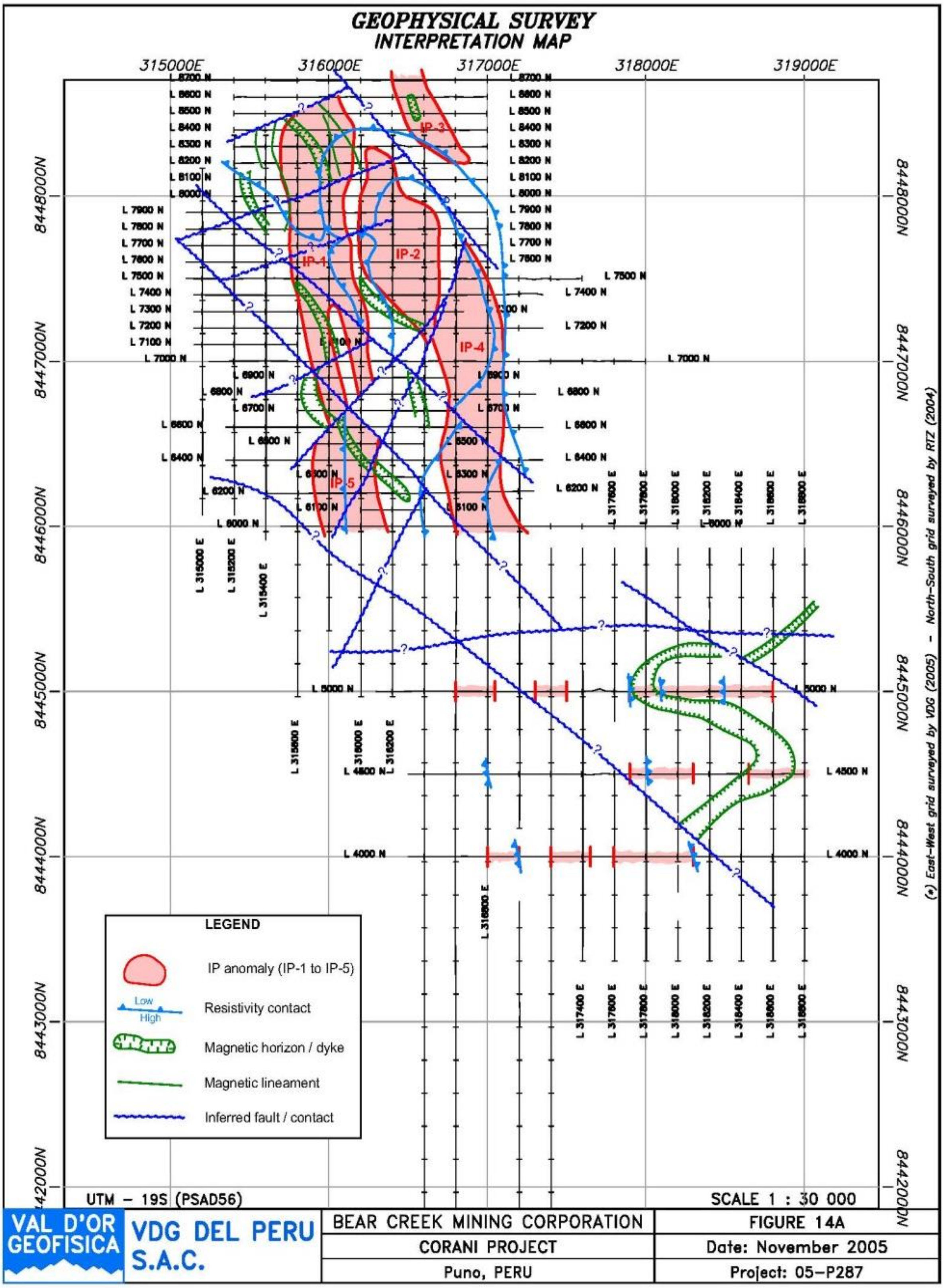


Figure 9-2 Geophysical Interpretation Map with Reconnaissance Area



9.2 Trenching Exploration

BCM has completed 25 trenches within the Project resource area (Corani Main, Minas, and Este) to verify the continuity of the structures covered by Quaternary sediments. Spacing between the trenches is roughly 50 to 100 meters. Channel samples from these trenches have produced an associated 1,295 assay intervals for a total of 2,924 meters of trench data. Another 16 trenches have been completed within the Project area south of the current resource area. All trenching and channel sampling is completed by hand, and samples are collected on 2-meter horizontal intervals based on GPS survey of the sample start and end points. Actual sample length is established by digitally draping the 2-meter sample intervals onto the topographic surface and adjusting the horizontal sample length to account for the recorded plunge or dip of the trench. In general, the results of trenching exploration indicate near surface mineralization both within and to the south of the current resource area.

9.3 Mapping

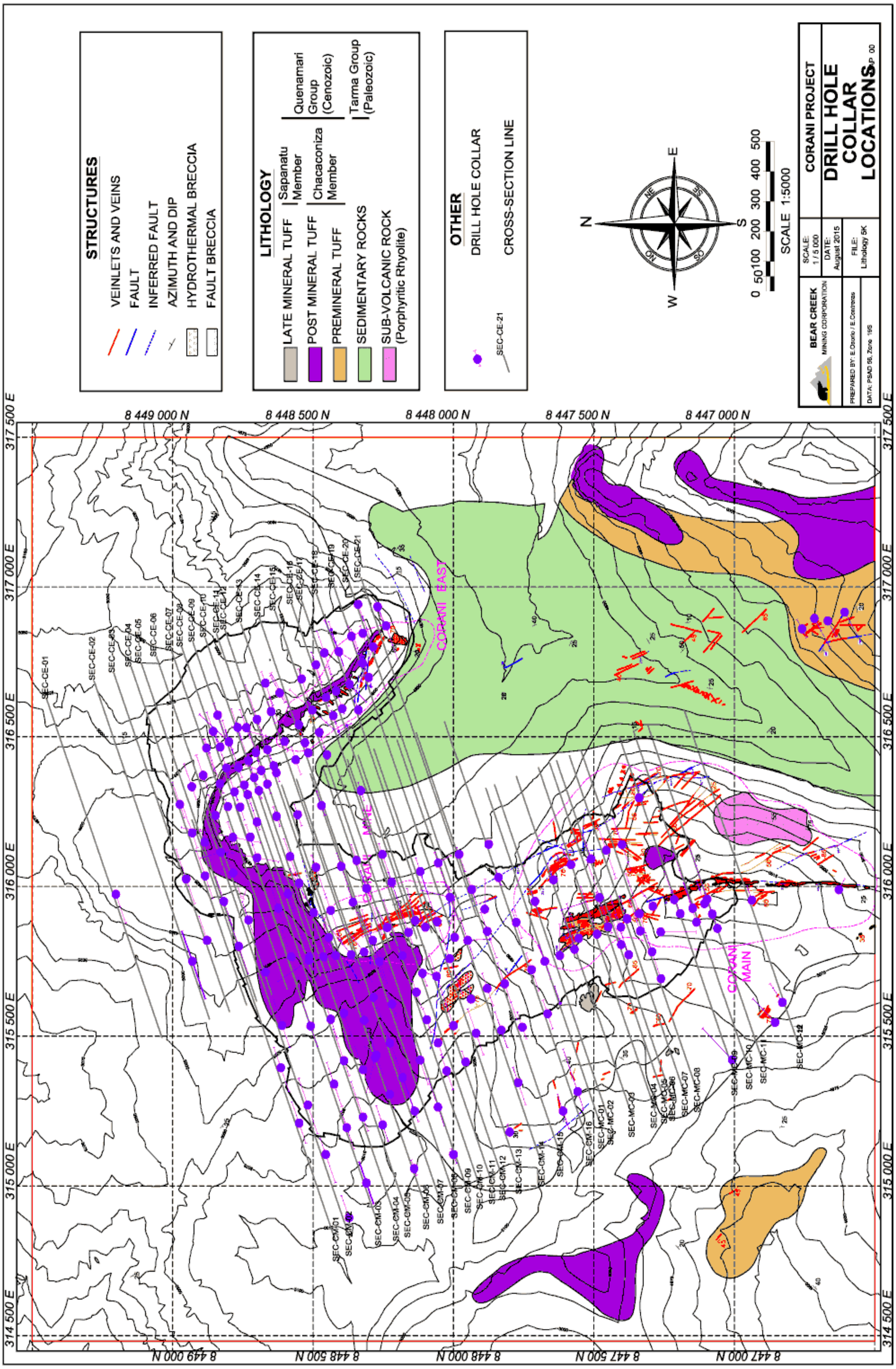
BCM has conducted general geologic surface mapping over the entire Project area. The total mapped surface is about 4.5 km wide (east-west) and 7.5 km long (north-south). In 2015, detailed surface mapping, including lithology, alteration, and structures, was performed at a scale of 1:2500 in the area of the proposed pits.

10 Drilling

10.1 BCM Drilling Exploration

Since 2005, BCM has completed a total of 556 drill holes at the Corani Project for a total of approximately 100,494.57 m. Drilling was completely by the Peruvian contractor, Bradly MDH primarily using LD250, JKS35, and LJ44 drill rigs. All of the drilling to date has been completed using diamond core drilling methods to produce either HQ (6.35 cm dia.) or NQ (4.76 cm dia.) core. Figure 10-1 shows the drill hole collar map that covers the area of the project with the estimated resource. Representative sections of the deposit are shown in Section 7.

Figure 10-1 BCM Drill Hole Collar Locations



The typical drill pattern employed by BCM consists of a series of drill fans on section lines spaced 50 m apart. The fans are arranged perpendicular to the strike of specific targets. Angled holes are used in an attempt to drill perpendicular to the orientation of the mineralization. Multiple holes are often drilled from one site in order to reduce surface impact and obtain the necessary drill coverage (sample spacing) at depth. Some areas have been in-filled to 25 m drill spacing and in other, largely lower-grade target areas, drill spacing remains at 100 m.

Drill hole collars were originally surveyed by hand held GPS with a reported accuracy of ±3.0 meters. Since 2008, all of the drill hole collars have been resurveyed by conventional survey with substantially higher precision. Comparison of the resurveyed collar elevations to the surface topography map indicated that about 15% of the collar elevations differed from the topographic elevation by more than 5 m. About two-thirds of these occur in steep topography where the collar elevation is below topography as a result of the cut required to establish the drill pad.

Prior to 2008, downhole surveys were not conducted. In 2008, downhole surveys were conducted on a series of 12 holes drilled targeting deep mineralized zones. Average depth of these holes is 220 m. Comparison of the downhole survey location of the bottom of the drill hole to the location projected from the collar survey revealed the average error of 4.89 m. The maximum of all errors was 11.13 meters. In the worst case, the drill hole location without survey would have been within one model block.

Based on the results of the surveyed vs. projected hole orientation comparison, the lack of downhole surveys for holes of 200 m or less should have no major impact on the development of the block model. Regardless, precision surface and downhole surveys are recommended for all future drilling exploration.

Core recoveries are generally excellent, with no discernible variation in rate of recovery between the two core sizes (HQ and NQ). While on site, the QP carefully reviewed the drilling and sampling procedures employed by BCM with BCM staff. Based on that review, the QP finds no drilling, sampling, or recovery factors that might materially impact the accuracy or reliability of the drilling results. The QP recommends that BCM produce annual (or seasonal) exploration reports to describe the drilling and sampling carried out during each given year or drilling campaign. The exploration report should contain adequate detail concerning the drill rig, drilling contractor, number of holes, total meters, recovery rates, drill targets and rationale for drill hole distribution, etc., to ensure that all pertinent information is captured and catalogued in a practical and efficient manner for ease of future use.

10.2 Drill Hole Data and General Drilling Results

Diamond drill hole data contained in the Project database to date includes 556 drill holes. Not all of the 556 drill holes included in the database are in the immediate area of the Project resource. The current block model relies on data from 470 drill holes, with an associated 34,443 sample intervals over a total of 85,212 m of drilling. The block model relies on 33,442 assay values each for silver, lead, zinc, and copper.

In general, drilling exploration has identified and further defined the distribution of mineralization within the three primary resource areas, Corani Main, Corani Este, and Corani Minas. Drilling results indicate that significant mineralization occurs in two basic forms; in large veins associated with the principal listric fault structures, and in stockwork veins found in the surrounding rocks. The larger veins are generally rich in quartz / silica and barite and contain typical higher lead-zinc-silver mineralization, and can be several meters in width and quite continuous along strike and down dip. The stockwork veins occur in the wall rock between the faults and also contain significant lead-zinc-silver mineralization. The basement sediments at Corani are largely barren, though local, discontinuous occurrences of mineralization have been encountered within these units.

Since 2006, BCM has undertaken to in-fill the known areas of mineralization in order to increase confidence in the resource estimate. This has been accomplished by in-fill drilling to produce a nominal 50-m drill hole spacing in previously more widely spaced drilling areas, with a focus on areas of higher grade mineralization.

Drilling exploration is summarized by year in Table 10-1.

Table 10-1: Exploration Drilling Summary by Year

Year	No. Holes	Resource Area	Comments/Results
2005	20	Este	Central and southern portion of deposit, directly above outcrop of mineralized structures
	20	Minas	In area of historic workings, intended to test the depth of known veins and breccias
	31	Main	Holes distributed along a wide, westerly dipping quartz-barite-sulfide structure
	3	Other	'Gold zone', south of Corani Main
2006	136	Este	Defined the form and continuity of Ag-Pb-Zn mineralization beneath post-mineral tuff
	115	Minas	Delineated the projection of mineralization in the north and east portions of the deposit
	43	Main	Holes drilled along the projection of the quartz-barite-sulfide strucucture and in areas of stockwork with silver sulfides
	39	Other	'Gold zone', south of Corani Main, defined the depth of the principal quartz-barite-oxide structure with a 14-m width and Au content ranging from trace to 11 gpt with 44 gpt Ag
	2	Other	Sedimentary units
2007	18	Este	Holes located in the north portion of the deposit and within the bofedal
	68	Minas	Drilling to confirm distribution of mineralization in the north and west portions of the deposit
	12	Main	Drilling targeted mineralization west of the principal structures
	17	Other	'Gold zone', south of Corani Main, drilling defined extent of mineralization to the north and at depth
2008	1	Este	
	2	Minas	
	3	Other	Sedimentary units, targeting potential mineralization to the south
2010	10	--	Metallurgical test holes
	10	Other	'Corani South', holes explored structures with Ag and Sb
2012	5	--	Geotechnical and condemnation test holes
	1	Other	'Corani South' exploration holes

11 Sample Preparation, Analyses and Security

The following description of BCM sampling and analytical procedures is based largely on details presented in the 2015 NI 43-101 Technical Report issued by BCM (M3, 2015), and in part on observations and conversation with BCM personnel during the QP site visit conducted in August 2017.

11.1 Sample Preparation

Diamond core samples are collected and placed into plastic and weatherproof cardboard boxes at the drill rig by the drill crew and are transported by vehicle to the Project camp, where the core preparation facilities are located. BCM geologists photograph the core as it is received from the drill rig and collect geotechnical (RQD) and core recovery information prior to selecting sample intervals for splitting. Assay samples, generally 2 m in length, are selected by the on-site BCM geologist and are split using a manual core splitter. One half of the sampled core is returned to the box for geologic logging, and the other half is bagged and tagged with a blind sample number assigned by BCM (Figure 11-1).

Figure 11-1 Bagged Core Samples Prepared for Shipment

Channel samples are collected by BCM geologists from hand-dug trenches using a hammer and a moil point chisel. The trenches are excavated by hand to remove the overburden and expose a clean bedrock surface on the trench floor.

Bagged trench and core samples are transported by BCM staff to Cusco or Juliaca, where they are transferred to a bus for shipment to (ISL-certified) ALS-Chemex labs in Arequipa, Peru. The samples are prepped in Arequipa, and are subsequently shipped to the ALS-Chemex lab in Lima for analysis. Chain of custody is documented throughout the entire transportation process.

The samples are prepared according to ALS-Chemex preparation code PREP-31, which entails the following:

- The sample is dried at 110° to120°C and crushed by jaw and roll crusher to 70% passing 2 mm (about #10mesh).
- A 250-g subsample is obtained using a riffle splitter.
- The split is pulverized using a ring-and-puck pulverizer to 85% passing 75 micron.
- Coarse rejects are returned to BCM.

11.2 Analytical Procedures

Silver, lead, zinc, and copper assays are carried out by three-acid digestion followed by atomic absorption spectrophotometry (AA) analysis according to ALS-Chemex method AA62, as outlined below:

- A sample of the pulp is digested with three acids: hydrofluoric, nitric, and perchloric, to produce a cake.
- The cake is leached with hydrochloric acid.
- The hydrochloric acid solution is subjected to AA to determine the concentration of dissolved silver, lead, zinc, and/or copper.

The procedure described above is reported to be robust over the reported range of 1 to 1,500 g/t silver.

Multi-element inductively coupled plasma (ICP) analysis is conducted on select sample intervals to assist with mineralization classifications and to guide the interpretation of the metallurgical process response.

11.2.1 Quality Assurance/Quality Control

BCM maintains an internal Quality Assurance/Quality Control (QA/QC) program which includes both standard and check sampling. The QA/QC program was initially limited to silver assay data. The check assays for lead and zinc began in 2011. BCM's QA/QC efforts currently do not include routine insertion of duplicate or blank samples into the sample stream, with the exception of a few gold blanks.

11.2.1.1 Standards

Commercially prepared standard samples are inserted into the sample stream at a rate of one standard for every 20 samples. Eight separate standard samples, each with a unique and specific certified assay value, are used. The standards are in pulp form, in contrast to the half-core samples from the Project, so are readily identifiable to the lab; however, the lab is not able to discern which specific standard has been inserted at any given time.

BCM personnel periodically review the standard sample analytical results. If the laboratory analytical result differs from the certified assay value by more than 10%, the entire associated assay run (set of 20 samples) is submitted for reassay.

11.2.1.1.1 2008 Standard Review

In 2008, Mr. Christian Rios, CPG, was reported to have conducted a critical review of BCM's QA/QC program. The work was previously reported in the 2015 NI 43-101 Technical Report issued by BCM (M3, 2015) and included the following discussion:

> As of March 2008, 20 sample batches had been submitted for reassay due to out of tolerance response of the associated standard. The author used the standards to check on assay bias and sample handling procedures.
>
> The statistical results of the ALS-Chemex silver assays indicated that the ALS-Chemex lab tended to undervalue the low-grade silver standards. A number of standards in the 1.2 to 1.9 gm/t range were reported back as trace by ALS. This is of no impact on reserves as the cut-off grades are significantly above these values.
>
> Several points were outside of the cluster of the data for that standard. These points are typically indications of sample swaps. There are probably about 0.5% sample swapping in the standards database. It is not known if this is a function of improper sample insertion, (the likely cause), or assay and database reporting errors elsewhere in the system.

11.2.1.1.2 2017 Standard Review

In 2017, GRE conducted a critical review of BCM's QA/QC program. Toward that end, BCM provided GRE with QA/QC data in multiple Excel spreadsheet files. GRE compiled the data into a single, comprehensive QA/QC data worksheet for analysis and evaluation. GRE prepared plots of the certified value of each standard against analytical values returned by the lab. This evaluation is used to check for calibration errors at the lab level and assay bias. Figure 11-2, Figure 11-3, and Figure 11-4 show the results of the standards check for silver, lead, and zinc, respectively. The line on the graph represents a 45-degree line, or the ideal result from all values. The silver analytical results generally correlate well with the standard values with a few outliers possibly indicating that the wrong standard was identified for the sample. The lead and zinc analytical results, however, correlate less well, with two sets of results showing over-valuation.

Figure 11-2 Corani Project Standards Results for Silver

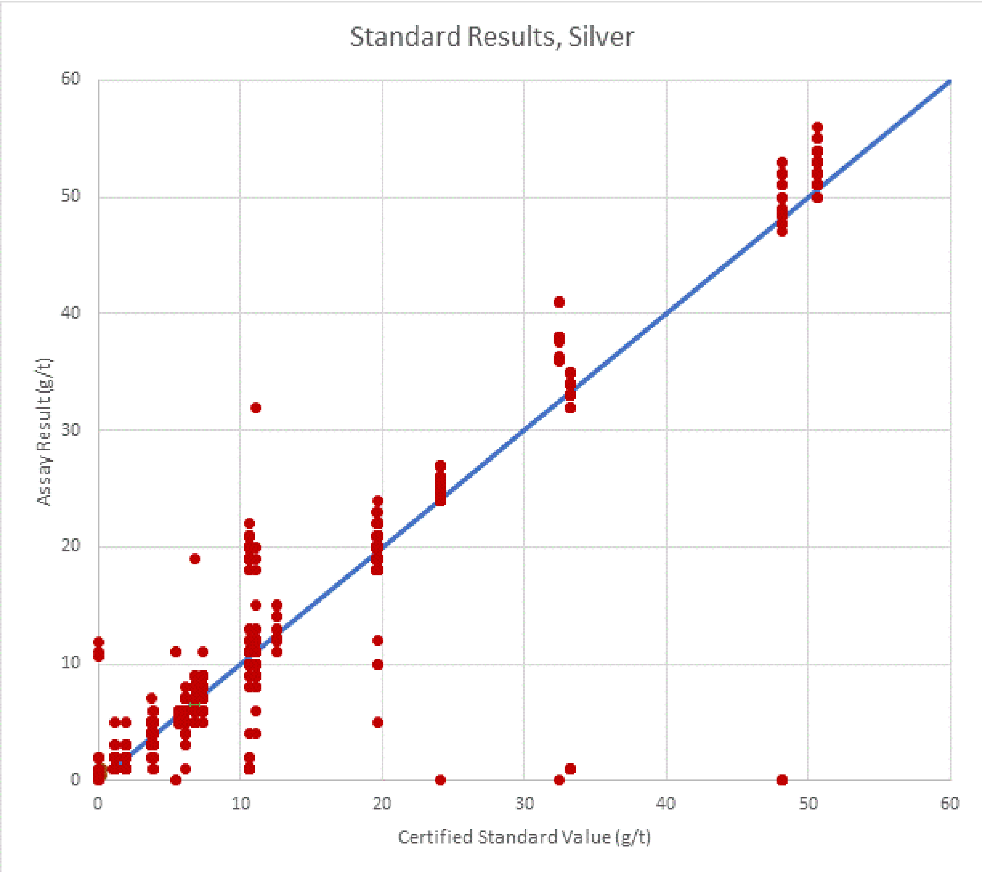


Figure 11-3: Corani Project Standards Results for Lead

Figure 11-4: Corani Project Standards Results for Zinc

11.2.1.2 Check Assays

11.2.1.2.1 2008 Check Assay Review

The following discussion was also prepared by Mr. Rios, and is presented here as previously reported in the 2015 NI 43-101 Technical Report issued by BCM (M3, 2015):

> Check assay pulps are submitted to a second laboratory on a roughly 1 in 25 basis. The initial check assay protocols were established during 2005 with silver assay checks only. The procedures were amended in late 2005 to include check assays of silver, lead, and zinc.
>
> The initial checks during 2005 are summarized on Figure 11-3. During this period, check assay pulps were submitted to SGS labs in Lima. There is a substantial variability in the check assay results during this period. There is no indication of bias in the data set, but there is substantial variability between the original and the check assays. The cause of the variability is not known. There is some potential that the check pulps have been swapped or mislabeled when shipped to the check lab.
>
> The same trend is apparent regarding the high degree of scatter with the Inspectorate checks as with the SGS checks before them. The variability occurs in all three metals, silver, lead, and zinc. Although a degree of scatter is typical for the precious metal assays, the variability of lead and zinc are unusual for base metal check assays on pulps.
>
> There is effectively no scatter in the plots for the most recent checks. It is not certain if the issue has been corrected or that it is not apparent with only 115 samples in the most recent check set.
>
> The variability in check assays for the period of 2005 through 2007 can be summarized by a quick scan of the percentage of checks that were more than 25% different than the original assay.

Silver	1978 checks	18.2% are more than 25% different
Lead	1983 checks	4.2% are more than 25% different
Zinc	1984 checks	7.3% are more than 25% different

> Hypothesis tests for each set of check assays do indicate that they can be accepted with 95% confidence and there is effectively no bias in the check assay result. However, the variability issue should be understood. It could simply be a function of miss-assignment of batch results to the working spreadsheet, or potential miss-labeling of pulps prior to shipment for check assay.
>
> Many of the scattered silver results are the same samples with scattered lead and zinc results. The implication is that the entire check assay has been mis-labeled, or mis-located when inserted into the master spreadsheet.

11.2.1.2.2 2017 Check Assay Review

In 2017, GRE conducted a critical review of the check assay results. GRE prepared quantile-quantile (or Q-Q) plots of the check assay data to determine if the two data sets come from populations with a common distribution. The Q-Q plot plots the quantiles (i.e., the fraction of points below a given value) of the first data set against the quantile of the second data set. A 45-degree line is also plotted. If the two sets come from a population with the same distribution, the points should fall approximately along the 45-degree reference line. A Q-Q plot can also test for shifts in location, shifts in scale, changes in symmetry, and the presence of outliers. Figure 11-5 through Figure 11-10 provide the check assay Q-Q plots prepared by GRE. Each plot also shows an R^2 value. The R^2 value is a statistical measure of how close the data are to the 45-degree regression line and can vary from 0 to 1. In general, the higher the R^2 value (i.e., the closer to 1), the better the model fits the data.

Figure 11-5: Inspectorate Check Assays (Silver) Q-Q Plot

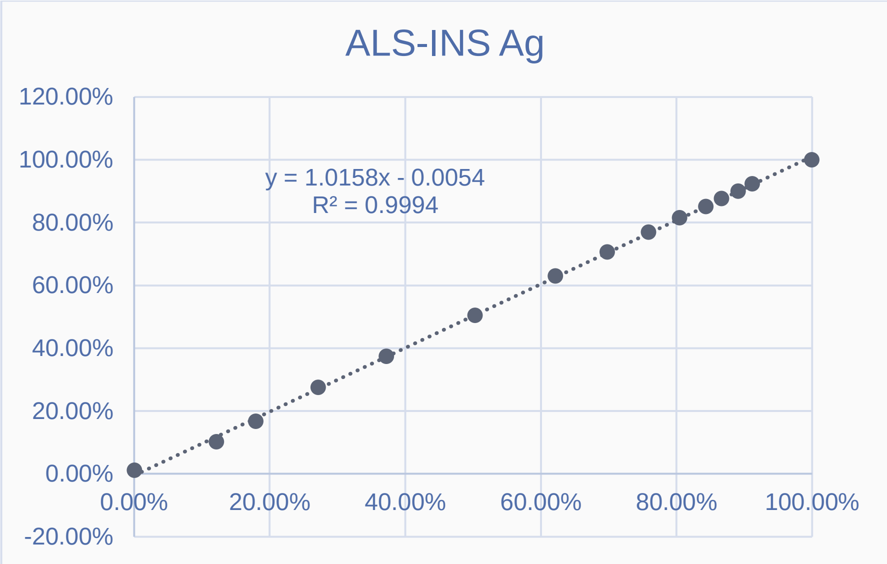


Figure 11-6: Inspectorate Check Assays (Lead) Q-Q Plot

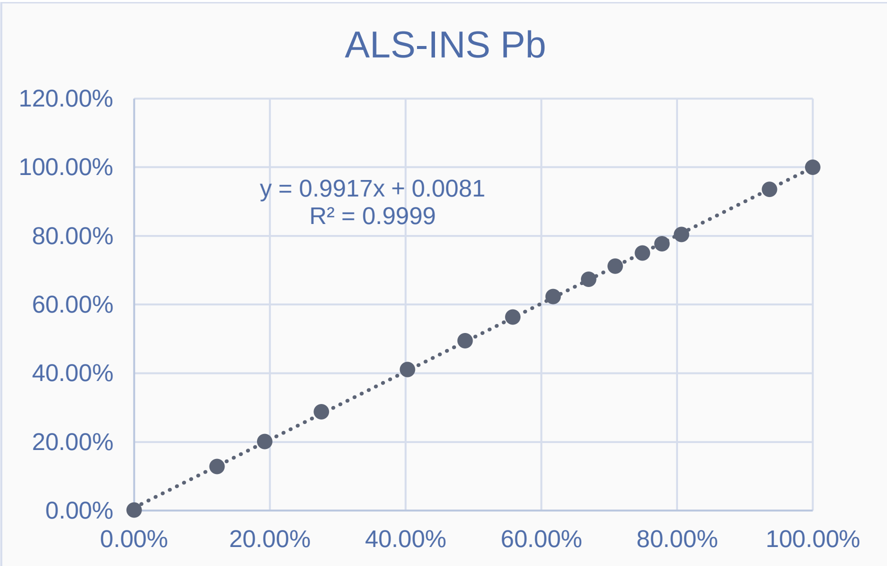


Figure 11-7: Inspectorate Check Assays (Zinc) Q-Q Plot

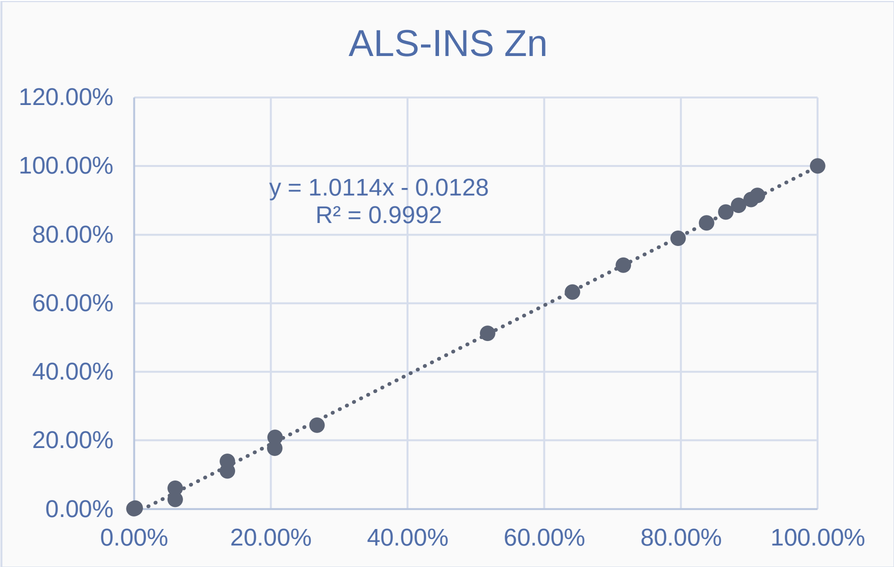


Figure 11-8: CIMM Check Assays (Silver) Q-Q Plot

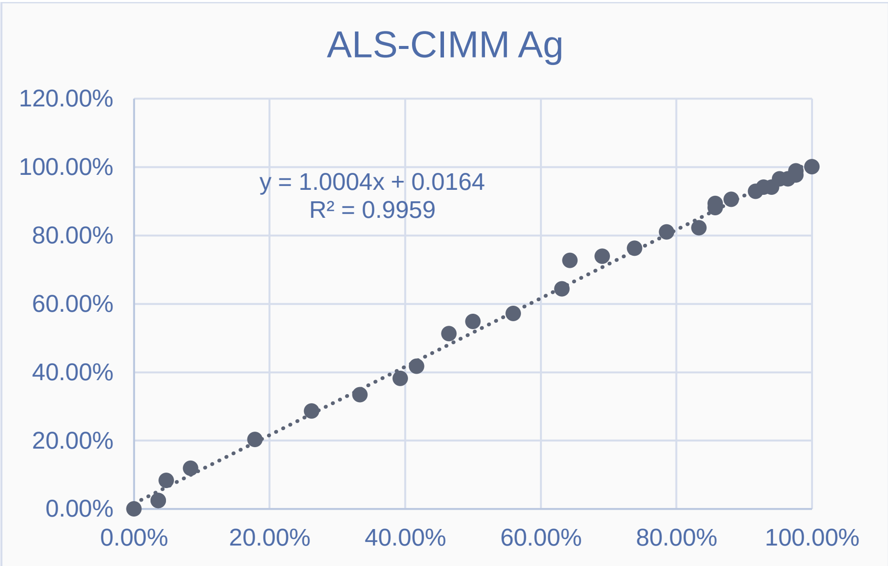


Figure 11-9: CIMM Check Assays (Lead) Q-Q Plot

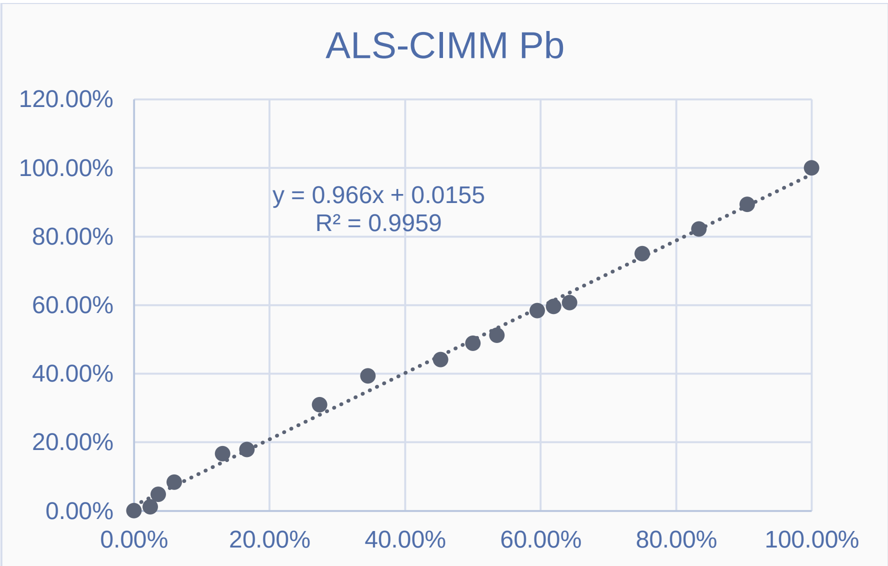
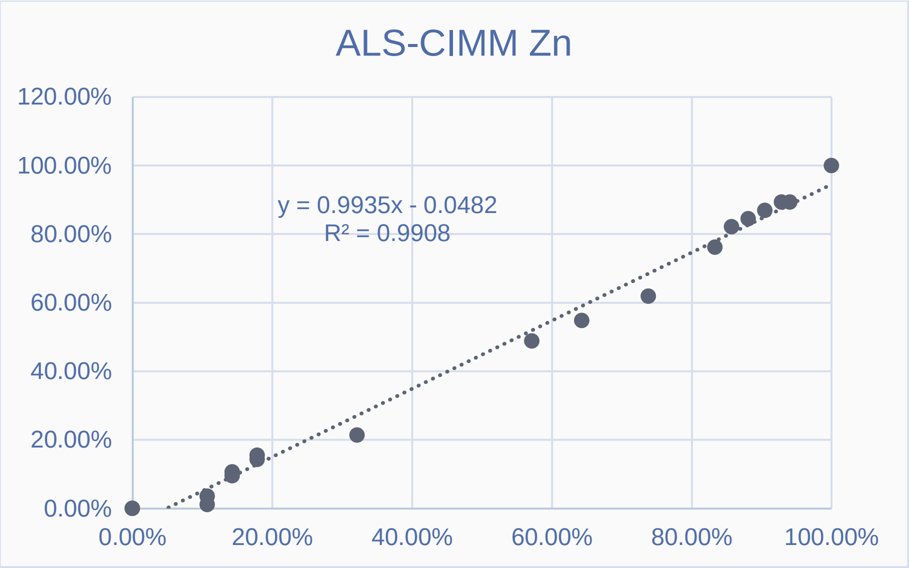


Figure 11-10: CIMM Check Assays (Zinc) Q-Q Plot



The Q-Q plots indicate effectively no scatter in the data, with R2 values ranging from a low of 0.9908 to a high of 0.9999. The ALS inspectorate Q-Q plots, in general show less scatter than the ALS CIMM Q-Q plots.

11.2.1.3 Blanks

Gold blanks were included for 16 samples in 2011. All gold analytical results for these 16 samples were below detection, indicating no artificially introduced contamination in the field for these samples.

Although BCM has not routinely inserted blank samples into the sample stream, the laboratories do periodic blank checks internally to check for trace sources of artificially introduced contamination within the lab. A cursory review of the lab blank results indicated no apparent bias introduced by the labs.

11.3 Sample Security

During exploration campaigns BCM employed standard chain of custody procedures during all segments of drill core and trench sample transport. Samples prepared for transport to the laboratory were bagged and labelled in a manner which prevents tampering, and remained in BCM control until released to private transport carrier in Cusco or Juliaca. Upon receipt by the laboratory, samples were tracked by a blind sample number assigned and recorded by BCM.

All whole and retained half core samples are stored in a fenced compound located in Juliaca (Figure 11-7). The core is neatly stored in labelled core boxes, which are arranged according to drillhole on sturdy, covered shelving units. Access to the core storage facility by unauthorized personnel is prohibited by a locked gate. Coarse reject and pulp samples are stored in BCM's Lima warehouse facility, which is accessible only to authorized BCM personnel.

Figure 11-11 Juliaca Core Storage Facility

11.4 QP Opinion on Adequacy

The QP finds the sample preparation, analytical procedures, and security measures described herein to be reasonable and adequate to ensure the validity and integrity of the data derived from BCM's sampling programs, with some room for improvement. Based on observations and conversation with BCM personnel during the QP site visit, in conjunction with the results of GRE's review and evaluation of BCM's QA/QC program, the QP makes the following recommendations:

- Formal, written procedures for data collection and handling should be developed and made available to BCM field personnel. These should include procedures and protocols for field work, geological mapping and logging, database construction, sample chain of custody, and documentation trail. These procedures should also include detailed and specific QA/QC procedures for analytical work, including acceptance/rejection criteria for batches of samples.

- A detailed review of field practices and sample collection procedures should be performed on regular basis, to ensure that the correct procedures and protocols are being followed.

- Review and evaluation of laboratory work should be an on-going process, including occasional visits to the laboratories involved.

- BCM's existing QA/QC program should be expanded to include at least standards, blanks, and duplicates. All QA/QC control samples sent for analysis should be blind, meaning that the laboratory should not be able to differentiate a check sample from the regular sample stream. The minimum control unit with regard to check sample insertion rate should be the batch of samples originally sent to the laboratory. Samples should be controlled on a batch by batch basis, and rejection criteria should be enforced. Ideally, assuming a 40-sample batch, the following control samples should be sent to the primary laboratory:

  - Two blanks (5% of the total number of samples). Of these, one coarse blank should be inserted for every 4th blank inserted (25% of the total number of blanks inserted).

  - Two pulp duplicates (5% of the total number of samples)

  - Two coarse duplicates (5% of the total number of samples)

  - Two standards appropriate to the expected grade of the batch of samples (5% of the total number of samples).

- For drill hole samples, the control samples sent to a second (check) laboratory should be from pulp duplicates in all cases and should include one blank, two sample pulps, and one standard for every 40-sample batch.

- The purpose of the coarse duplicates is to quantify the variances introduced into the assay grade by errors at different sample preparation stages. Coarse duplicates are inserted into the primary sample stream to provide an estimate of the sum of the assay variance plus the sample preparation variance, up to the primary crushing stage. An alternative to the coarse duplicate is the field duplicate, which in the case of core samples, is a duplicate from the core box (i.e. a quarter core or the other half core). Because sample preparation is currently carried out by the laboratory (and not by BCM), if coarse duplicates are preferred (in order to preserve drill core), the coarse duplicates should be sent for preparation and assaying by the second laboratory.

- QA/QC analysis should be conducted on an on-going basis, and should include consistent acceptance/rejection tests. Each round of QA/QC analysis should be documented, and reports should include a discussion of the results and any corrective actions taken.

12 Data Verification

12.1 Site Inspection

GRE representative and QP J.J. Brown, P.G., conducted an on-site inspection of the Corani Project and the Juliaca core storage facility on August 7 through 10, 2017. Ms. Brown spent two full days at Project site accompanied by BCM geology staff Enrique Osorio Cornejo and Jorge Ganoza. While on site, Ms. Brown conducted general geologic field reconnaissance, including inspection of bedrock exposures and other surficial geologic features, ground-truthing of reported drill collar and trench sample locations, and superficial examination of historic mine workings. One full day of the site visit was spent at the core storage facility in Juliaca, where select intervals of whole and half core were visually inspected and samples were selected to submit for check assay.

Field observations during the site visit generally confirm previous reports on the geology of the Project area. Bedrock lithologies, alteration types, and significant structural features are all consistent with descriptions provided in existing Project reports, and the author did not see any evidence in the field that might significantly alter or refute the current interpretation of the local geologic setting (as described in Chapter 7 of this report).

A total of 96 collar locations (approximately 20% of the total used to develop the block model) were verified in the field using a hand-held GPS unit. The average variance between field collar coordinates and collar coordinates contained in the Project database is roughly 18 m, well within the expected margin of error accounting for the difference between the methods of survey (hand held unit versus professional ground survey).

Specific core intervals from 35 separate drill holes were selected for visual inspection and potential check sampling based on a preliminary review of the drill hole logs and associated assay values. The core intervals were selected prior to the site visit, and the core was laid out by BCM staff and ready for inspection upon arrival. With few exceptions, the core samples accurately reflect the lithologies recorded on the logs. Given the similarity in composition of the pre- and post-mineral tuff, it is often difficult to distinguish between the two without greater context. In some cases, (unsampled) core intervals logged as post-mineral tuff were variably altered and oxidized, similar in character to the majority of the core logged as pre-mineral tuff. Three such sample intervals were selected from a single drill hole, DDH-C59B, for assay. The sample intervals selected were gradational with regard to visible evidence of alteration/oxidation (i.e., one very altered, one moderately altered, and one barren in appearance). Assay results (Table 12-1) indicate that the samples are in fact mineralized.

Table 12-1: Previously Unsampled Postmineral Tuff Assay Results

			Assay Results		
	Interval		Ag	Pb	Zn
DHID	From	To	ppm	%	%
DDH-C59B	11.67	12.81	66	0.25	0.03
DDH-C59B	26.01	27.33	141	2.19	0.07
DDH-C59B	32.67	33.87	35	0.14	0.01

A total of 17 samples were selected for check assay. The samples were selected from low, moderate, and high-grade intervals based on original assay results. In all cases, the degree of visible alteration and evidence of mineralization observed was generally consistent with the grade range indicated by the original assay value. Laboratory analysis was completed by ALS Peru using the same sample preparation and analytical procedures as were used for the original samples. Standard t-Test statistical analysis was completed to look for any significant difference between the original and check assay population means. A single sample was removed from the total sample population based on an erroneous original assay value. The results of the t-Test showed no statistically significant difference between the means of the two trials (original versus check assay).

12.2 IMC Audit

In 2011, BCM contracted Independent Mining Consultants to perform an audit of the digital Project database. The following discussion was previously presented in the 2015 NI 43-101 Technical Report issued by BCM (M3, 2015):

IMC compared a random selection of original assay certificates to the assay information contained in the Corani Project database. Assay data from the following drill holes was used during the audit process:

DDH-C3-A	DDH-C7	DDH-C12-A	DDH-C16-B	DDH-C18-A	DDH-C20-A
DDH-C29-B	DDH-C32-A	DDH-C34-A	DDH-C41	DDH-C42-A	DDH-C43-B
DDH-C46-A	DDH-C58-B	DDH-C66-A	DDH-C70-A	DDH-C74-B	DDH-C79-A
DDH-C84-A	DDH-C86-A	DDH-C92			

The assay certificate data was entered into an excel spreadsheet and then added to the IMC database containing the Corani data.

The 21 drill holes evaluated had 1,524 associated silver, copper, lead, and zinc assay intervals. Assay certificate data was received for 1,310 assay intervals; the intervals for which original assay data was not provided are summarized in Table 12-2.

Table 12-2: Missing Original Assay Certificates

Hole Number	Interval	Number of Intervals Missing Assay Certificates
DDH-C3-A	1,988 to 2,052	62 intervals
DDH-C12-A	2,210 to 2,287	74 intervals
DDH-C16-B	9,388 to 9,447	57 intervals
DDH-C20-A	4,669 to 4,690	21 intervals

Thirty-one silver assays with a value less than 1.0 gm/t were entered into the database with a value of 1.0. Otherwise, there were 2 assay intervals with silver assay values that did not match the assay certificates (Table 12-3).

Table 12-3: Certificate Check Errors for Silver

Hole Number	FROM (meters)	TO (meters)	Sample Number	Database Grade gm/t	Certificate Grade gm/t
DDH-C41	98	100	8842	8	108
DDH-C70-A	104	106	15521	29	11

Eight lead assays with a value less than 0.01 were entered into the database with a value of 0.01. There were 2 assay intervals with lead assay values that did not match the assay certificates (Table 12-4).

Table 12-4: Certificate Check Errors for Lead

Hole Number	FROM (meters)	TO (meters)	Sample Number	Database Grade %	Certificate Grade gm/t
DDH-C41	98	100	8842	0.04	5.90
DDH-C70-A	104	106	15521	0.02	0.01

Two zinc assays with a value less than 0.01 were entered into the database with a value of 0.01. There were 2 assay intervals with zinc assay values that did not match the assay certificates (Table 12-5).

Table 12-5: Certificate Check Errors for Zinc

Hole Number	FROM (meters)	TO (meters)	Sample Number	Database Grade %	Certificate Grade %
DDH-C41	98	100	8842	0.01	1.59
DDH-C70-A	104	106	15521	0.04	0.01

A total of 383 copper assays with a value of less than 0.01 were entered into the database with a value of 0.01. There was 1 assay interval with a copper assay value that did not match the assay certificate (Table 12-6).

Table 12-6: Certificate Check Errors for Copper

Hole Number	FROM (meters)	TO (meters)	Sample Number	Database Grade %	Certificate Grade %
DDH-C41	98	100	8842	0.01	0.13

The results for the individual metals all appear on the same record, indicating that two records out of 1,310 records were entered in error. This sampling shows an error rate of 0.15%, which is an acceptable error rate, and the data is considered reasonably accurate and suitable for use in estimating mineral resources and mineral reserves.

The observed discrepancy between trace assay entries and certificate values is likely a function of continuity between data entry personnel. This issue is minor and has no material impact on the determination of mineral resources or reserves, but should be addressed for consistency. The stated procedure by BCM personnel is to enter the less than trace results at half of the value of the trace assay. For example, <1 gm/t silver should be entered into the database as 0.5 gm/t silver according to BCM protocol.

IMC also compared trench sample assay values with nearby core sample assay values. IMC composited the data into 8-m down hole (or down trench) length composites. The composites were then paired on a nearest neighbour basis.

The nearest neighbour procedure finds pairs of trench and diamond drilling composites that are within a specified distance to each other, so that a statistical comparison of the two data sets can be completed. For this test, IMC used 8-m, 16-m, and 24-m spacing between data pairs, which corresponds to the unit size of the 8-m length composites. There were only 20 to 25 pairs at the 8-m spacing, but there were over 80 pairs at the 16-m spacing, which is a sufficient quantity to provide for a robust statistical estimate.

Table 12-7 summarizes the results of the nearest neighbour comparison. The comparison was applied for silver and lead only, as trench samples were not assayed for zinc. Statistical hypothesis tests were completed on the two closely located sample sets. The pass/fail determinations presented in Table 12-7 are based on the application of a 95% confidence interval. The T-test is a comparison of the population means. The Paired T calculates the differences between individual pairs and confirms that the differences are sufficiently small. The binomial test is a check of how many times one population is greater than the other, and the KS (Komologorov-Smirnoff) test is a comparison of the overall shape of each distribution.

In all cases, the test results indicate that the trench and core sample assay data can be commingled for the purposes of developing a block model and calculating the mineral resource estimate.

Table 12-7: Nearest Neighbor Comparison – Trench vs. Diamond Drill Samples

Metal	Maximum Spacing Between Composites	Number of Pairs	Diamond		Trench		T Test on Means	Paired T on Pairs	Binomial Test	KS Test
			Mean	Variance	Mean	Variance				
Silver (gm/t)	16 meters	84	82.10	3534.6	82.94	3622.2	Pass	Pass	Pass	Pass
Lead (%)	16 meters	87	1.00	0.439	1.09	0.621	Pass	Pass	Pass	Pass

12.3 GRE Audit

12.3.1 Digital Project Database

In 2017, GRE completed a QAQC audit of the digital Project database. GRE compared a random selection of original assay certificates to the assay information contained in the Corani Project database. Assay data from the following certificates was used during the audit process:

LI05099746	LI06037967	LI07052871	LI07003983	LI08034674	LI05029718
LI05074420	LI07032682	LI05023340	LI06040456	LI06115686	LI06007458
LI07026517	LI06036875	LI06080182	LI07041835	LI06115686	LI06047343
LI07026518	LI06037966	LI06058508	LI07063684	LI07038585	

The results of the audit are summarized below:

- All assay results viewed that were less than the reporting limit were entered into the database with a value of 0.
- Samples from six of the checked certificates were not located in the Project database.
- One certificate included sample IDs that were in the database twice, with one set of results consistent with the certificate and one set different from the certificate.
- All other results were consistent between the database and the certificates.

The observed discrepancy between trace assay entries and certificate values is likely a function of continuity between data entry personnel. This issue is minor and has no material impact on the determination of mineral resources or reserves, but should be addressed for consistency. The stated procedure by BCM personnel is to enter the less than trace results at half of the value of the trace assay. For example, <1 gm/t silver should be entered into the database as 0.5 gm/t silver according to BCM protocol.

The other QA/QC checks show minimal to no error, which GRE believes represents an acceptable error rate, and the data is considered reasonably accurate and suitable for use in estimating mineral resources and mineral reserves.

GRE also completed a mechanical audit of the Project database in order to evaluate the integrity of data from a data entry perspective. The mechanical audit identified a small number of data entry errors, including gaps, overlaps, and missing sample intervals. All data entry errors were easily rectified, and are considered insignificant with regard to potential impact to the mineral resource and mineral reserve estimates.

12.3.2 Density Data

BCM has performed 1,100 density determinations using a waxed core method. Samples are chosen out of every 5th core box to provide a sample spacing of approximately 15 m.

In 2015, GRE analyzed the density data to determine if a relationship exists between density and a variety of parameters, including assay grade, mineralization type, and deposit area. Silver assay values in gm/t were converted to percent to normalize the grade units. Statistical analysis indicates that the best predictor of density is a combined silver, lead, and zinc grade of 0.9381%. The average density of samples with a combined grade less than 0.9381% is 2.31 t/m^3, and the average density of samples with a combined grade of greater than or equal to 0.9381% is 2.43 t/m^3. The average density of post-mineral tuff with no associated assay data is 2.3 t/m^3, and the average density of non-tuff material is 2.53 t/m^3. The densities (in t/m^3) applied for each lithologic unit during mineral resource estimation are presented in Table 12-8.

Table 12-8 2015 Updated Densities

Rock Type	Grade	Density
Pre-Mineral Tuff	< 0.9381	2.31
Pre-Mineral Tuff	>= 0.9381	2.43
Post-Mineral Tuff	Not Applicable	2.3
Other Materials	Not Applicable	2.53

12.4 QP Opinion on Adequacy

Based on the results of the QP's check sampling effort, verification of drill hole collars in the field, visual examination of selected core intervals, and the results of both manual and mechanical database audit efforts, the QP considers the collar, lithology, and assay data contained in the Project data base to be reasonably accurate and suitable for use in estimating mineral resources and reserves.

The database audit work completed to date indicates that occasional inconsistencies and/or erroneous entries are likely inherent or inevitable in the data entry process. The QP recommends that BCM establish a routine, internal mechanical audit procedure to check for overlaps, gaps, total drill hole length inconsistencies, non-numeric assay values, and negative numbers. The internal mechanical audit should be carried out after any significant update to the database, and the results of each audit, including any corrective actions taken, should be documented and stored for future use in database validation.

Based on the positive assay results of the selected intervals of previously unsampled post-mineral tuff, the QP recommends that BCM sample at least one 2-m interval for every 20-m drilled and logged as post-mineral tuff. If positive assay results are returned, additional intervals should be selected accordingly to ensure that all mineralized material is analyzed.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

The reader should note that Sections 13.2 through 13.6.3 describe the work relating to the 2011 Feasibility Study and Sections 13.6.4 through 13.7.2 describe the work associated with the 2015 Optimized and Final Feasibility Study Feasibility Study. No metallurgical work has been undertaken since then.

Metallurgical testing of the Corani deposit has been extensive. Over one hundred samples have been tested using a variety of testing protocols including bulk flotation, cyanidation and sequential flotation. Metallurgical response to these conventional metal recovery methods was, however, widely variable. Some samples did not respond to conventional flotation recovery, with bulk flotation being necessary. Others produced clean concentrates at relatively good recoveries. The 2015 Feasibility Study refined these results and provided clarity for a predictable metallurgical recovery model which can be used in conjunction with the block model for mine planning.

The high degree of variability was attributed to the mineralogy of the deposit. Many of the poorer performing samples had high degrees of non-sulfide lead, resulting in poor lead recovery and zinc selectivity issues during lead flotation. In addition, the mineral grain size of many of the samples was very fine, and interlocking with other sulfides would make recovery of high-grade concentrates at high recovery very challenging.

While geological classification has allowed definition of some of the zones by metallurgical response, accurate prediction of FBS mineralization metallurgical performance had remained elusive. However, based on the 2015 study, much of the necessary data for predicting recoveries in a detailed block model are contained in the existing database. The following describes original and subsequent metallurgical work performed by GRE and TS leading to the current understanding of metallurgy and recoveries.

13.2 Samples

A plethora of testing, on a range of samples, has been conducted across the Corani deposit. In summary, about 66 discrete sections of drill core were batch tested using a sequential lead-zinc flotation flowsheet. Closed circuit metallurgical performance was determined via completion of locked-cycle tests on a total of 16 samples. Bulk flotation response, evaluated via batch flotation tests, was determined for an additional ± 70 samples. Notably, for a portion of these samples, zinc contents were low and only lead concentrate was produced. The response of 46 of the samples to cyanidation leaching was tested on whole ore. In a single test, cyanidation of a flotation tailing was evaluated. The majority of the samples tested in the flotation evaluations, bulk and sequential, were subjected to mineralogical assessment, primarily using QEMSCAN. A summary of tests conducted by year and testing facility is summarized in Table 13-1.

Table 13-1: Number of Discrete Samples for Metallurgical Testing

Test Type	Dawson 2006	G&T 2007	SGS 2007/2008	SGS 2009	SGS 2011	Total
Sequential Batch	5	6	31	-	25	67
Sequential Cycle	1	-	12	2	2	17
Bulk	8	71	-	-	-	79
Whole Ore CN	12	32	2	-	-	46

A description of the geological classification used by BCM to delineate the samples is as follows:

- CSC - Coarse-grained silica-sulfide-celadonite characterized by readily discernible sulfides (galena-sphalerite-chalcopyrite+-tetrahedrite) with celadonite in crystalline to locally opaline quartz;

- CS - A subset of CSC that contains coarse galena-sphalerite-chalcopyrite +- tetrahedrite without green celadonite clay;
- TET – Ag-bearing tetrahedrite characterized by recognizable late-stage, coarse-grained tetrahedrite cutting earlier sulfides and displaying the highest Ag contents: normally ores with low Pb-Zn contents;
- PM – Pyrite-marcasite +- quartz typical of early-stage mineralization with little polymetallic mineralization;
- FBS – Fine-grained black silica-sulfides characterized by very fine-grained mineralogy deposited from quenched ore fluids with highly variable metal content;
- QSB - Crystalline quartz-sulfide-barite interpreted as early fault fill or late-stage breccia fill;
- PG – Plumbogummite, identified as a pale-green, waxy, Pb-phosphate mineral that in metallurgical test results showed diminished lead flotation and difficulties in separation of base metals;
- FeOx – Iron-oxide mineralization with locally elevated Ag and generally low Pb-Zn. This is a gradation zone with mixtures of FeOx and FBS; and
- MnO – Manganese-oxide mineralization hosting mainly Ag with lesser Pb-Zn.

A summary of the number of samples tested for each geological classification is provided in Table 13-2. FBS is understood to be the main geological lithology by tonnage in the Corani deposit, and thus, the samples were also dominated by this geological classification.

Table 13-2: Number of Samples Tested by Ore Type

Ore Type	Dawson 2006	G&T 2007 Bulk Tests	G&T 2007 CN Tests	SGS 2007/2008	SGS 2009	SGS 2011
CSC	2	6	-	2	-	-
FBS	2	26	8	17	-	11
PM	-	15	3	6	-	4
QSB	-	7	3	2	-	6
FeOx	1	4	5	1	-	2
MnO	-	6	6	2	-	1
PG	-	7	7	-	-	1
Mixed	5	-	-	-	2	8

Notes: a) Samples that contained two ore types were categorized into the dominant ore type for that sample.
b) Samples from which samples across multiple holes were combined are referred to as ‘mixed’.

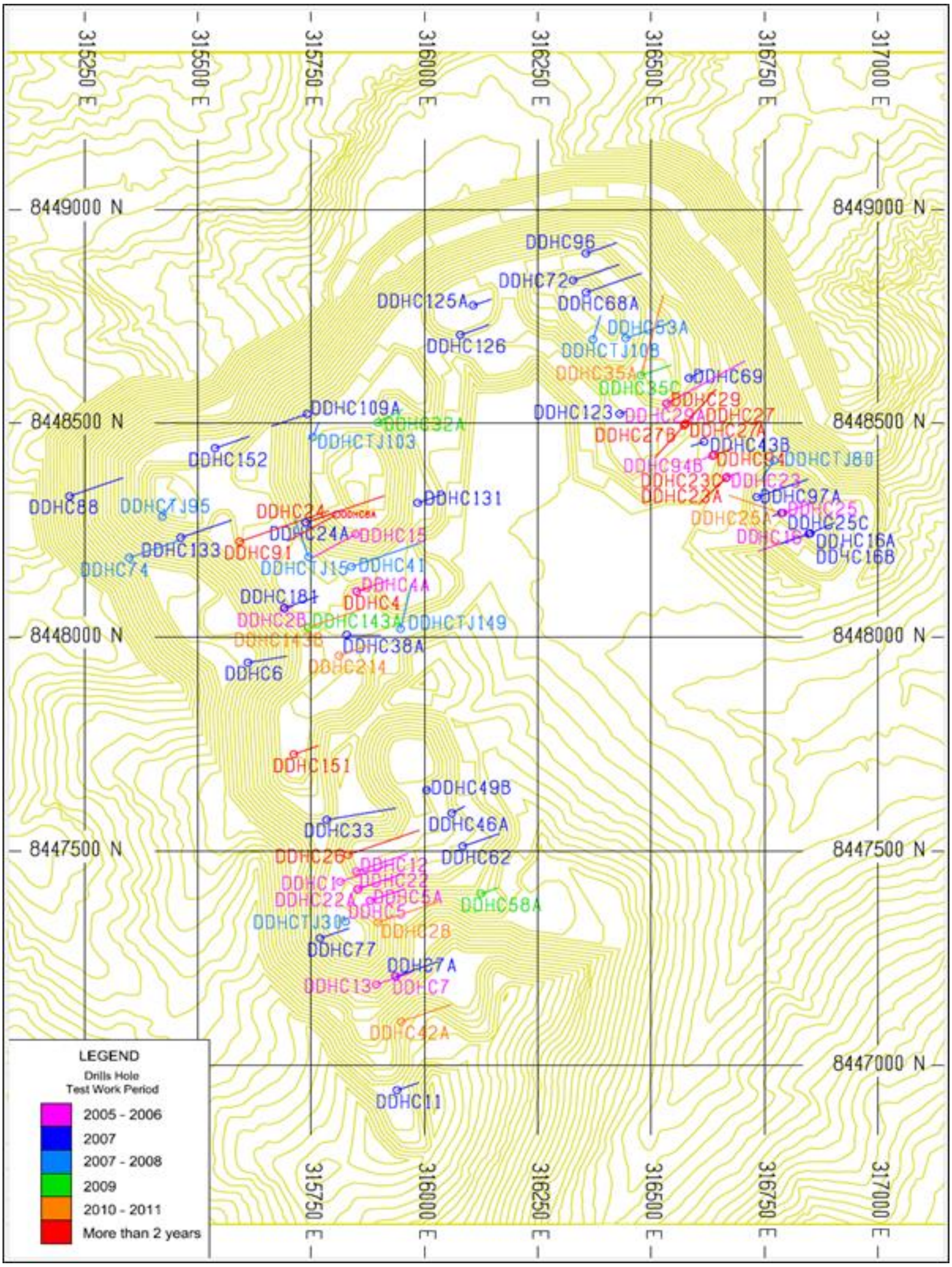
In total, 36 samples were subjected to ore hardness testing using different grindability testing protocols. A summary of the number of samples tested using the different comminution tests is summarized in Table 13-3.

Table 13-3: Number of Comminution Tests Conducted on Discrete Samples

Test / Deposit	SGS Vancouver 2008	SGS Chile 2010	SGS Chile 2012
SPI	10	6	20
Bond Ball	10	6	20
Bond Abrasion	2	6	20
SMC Test	-	6	20
Bond Rod	-	6	20
LEIT	-	6	20
PLT	-	-	17
UCS	-	-	13
Este	2	2	11
Minas	6	2	7
Main	2	2	2

The location of the metallurgical testing and grindability samples are shown in Figure 13-1.

Figure 13-1: Location of Drill Holes from which Metallurgical Test Samples were sourced.



13.3 Mineralogy

Lead occurs not only as Galena, but also in non-sulphide forms such as plumbogummite ($PbAl_3(PO_4)2(OH)5$•(H_2O)). Non-sulphide forms of lead would not be expected to be recoverable by flotation. According to the SGS dataset, generated using QEMSCAN technology, galena and sphalerite are both very finely grained. Fine primary and regrind sizes would be anticipated to facilitate separation of the two minerals into clean flotation concentrates.

The relationship between these mineralogical characteristics and metallurgical response was demonstrated in Figure 13-2 through Figure 13-4, prepared by SGS, relating the mineralogical parameters to bulk flotation response. In these figures, SGS related the QEMSCAN mineralogical data that they generated against the metallurgical performance data generated by G&T using a bulk circuit for the same samples. Figure 13-2 displays the relationship between galena grain size and bulk circuit lead recovery. This data indicates galena grain size is an important driver of metallurgical response. Also of note is that the grain sizes noted are very fine, measuring less than 17µm in all cases.

Figure 13-2: Relationship between Lead Recovery and Galena Grain Size (graph from SGS)

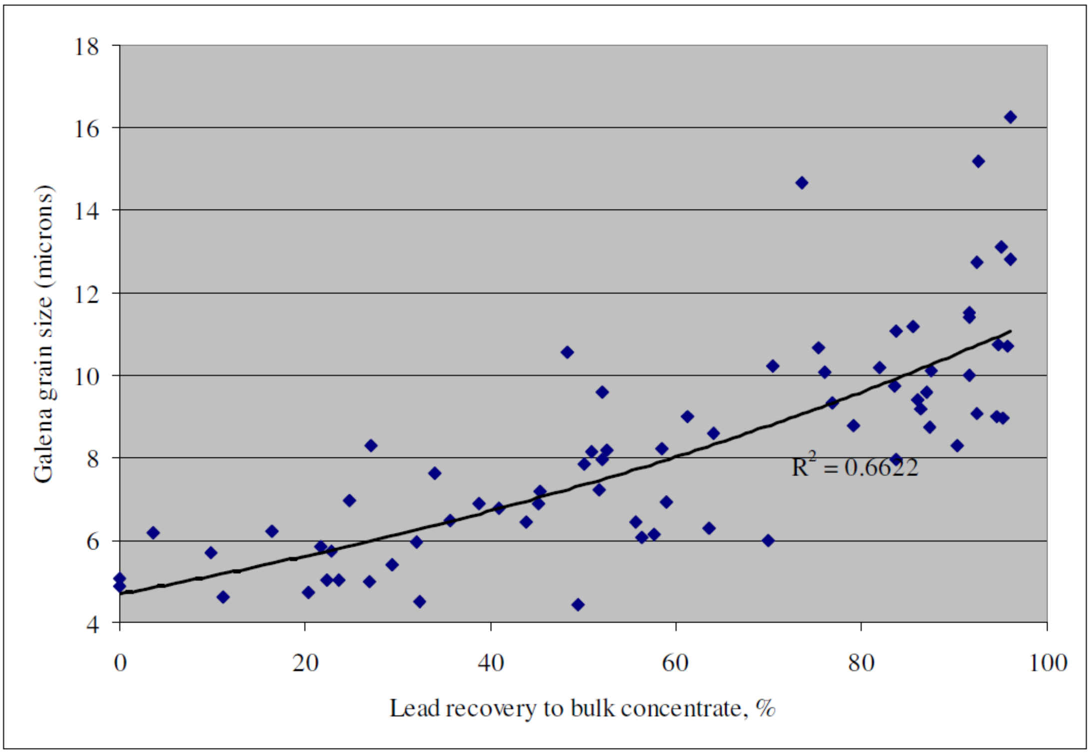
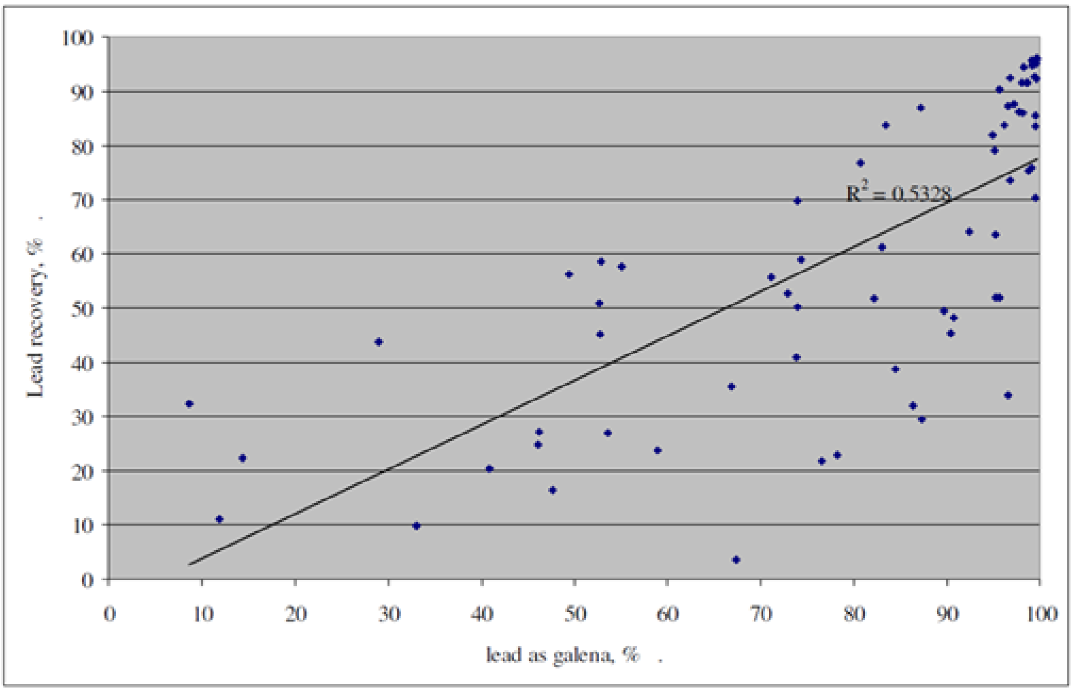


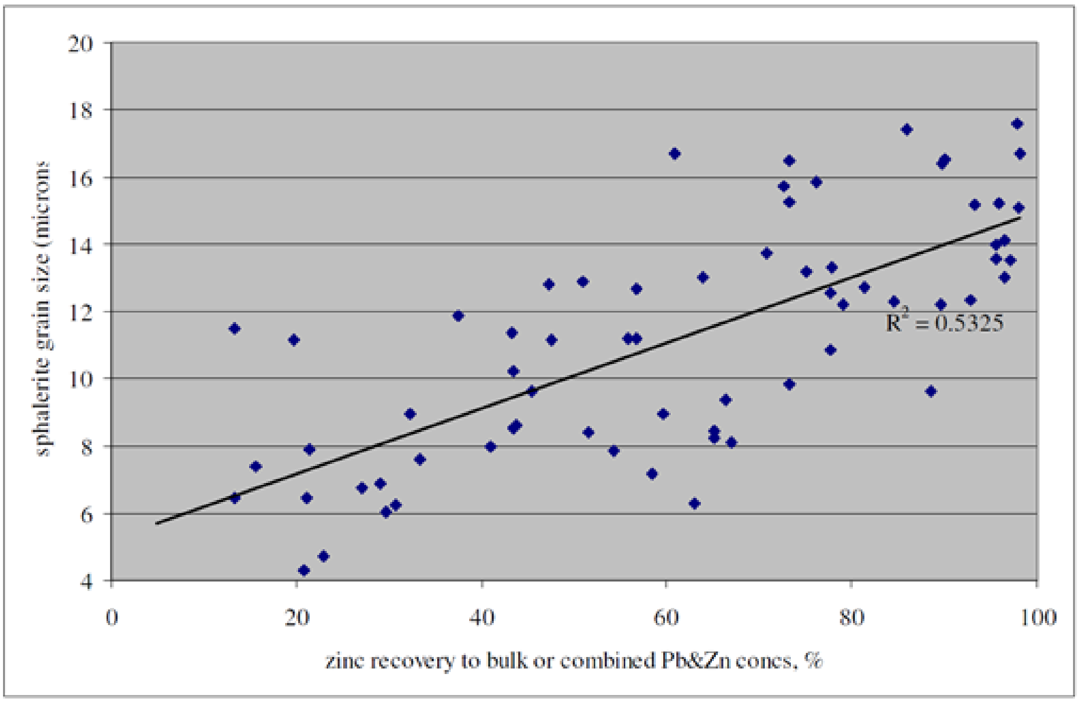
Figure 13-2 displays the correlation between the deportment of lead to galena and lead recovery to the bulk circuit for the same samples as Figure 13-3. The relationship is less clear than for galena grain size but indicates that this deportment plays a role in lead recovery. This is not surprising as non-sulfide forms of lead are not generally recoverable by flotation.

Figure 13-3: Relationship between Lead Recovery and Deportment of Lead to Galena (graph from SGS)



Zinc present in the samples was identified as sphalerite. Zinc performance was found to be mainly driven by sphalerite grain size. The relationship between zinc recovery and sphalerite grain size is shown in Figure 13-4. The grain sizes were similarly fine as for galena, showing the need for fine primary and / or regrinding.

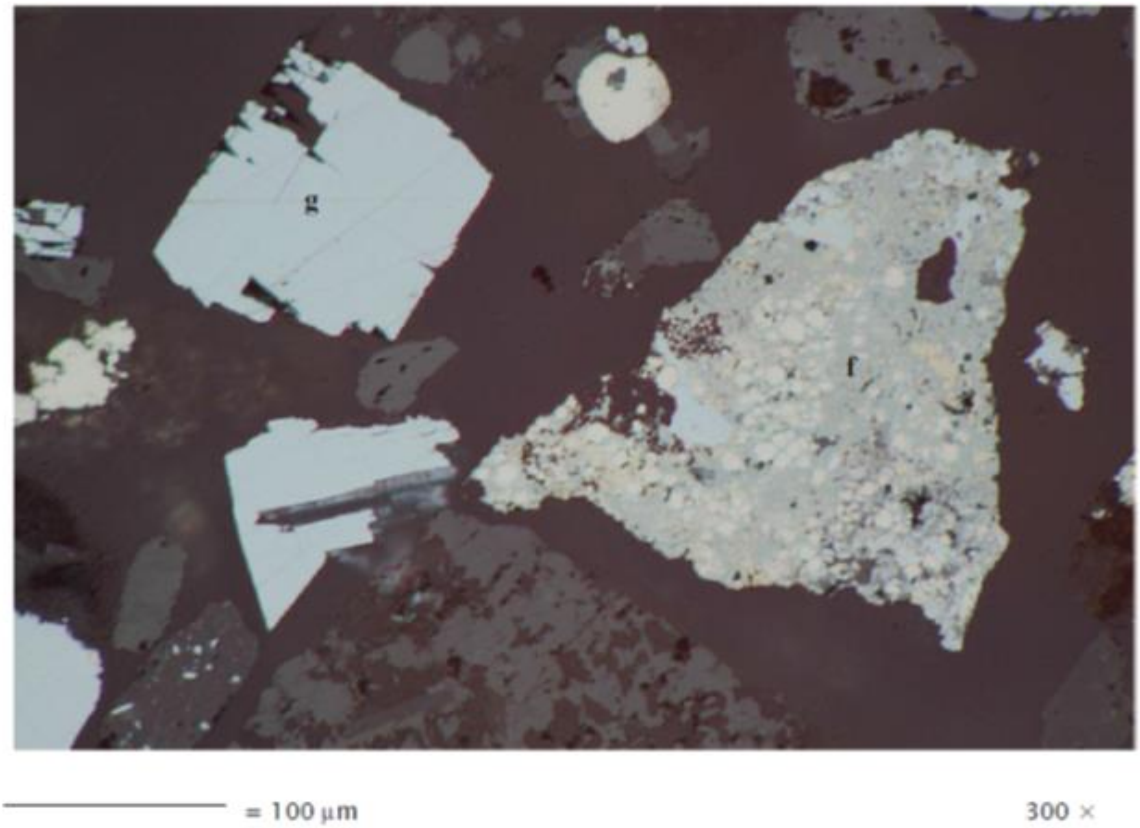
Figure 13-4: Relationship between Zinc Recovery and Sphalerite Grain Size (graph from SGS)



Silver mineralogy was briefly investigated by Hazen research and then again by SGS in 2008. The data indicated that silver was present mainly as tetrahedrite. Other sulfosalts were identified as minor carriers and included: myargyrite, pyrargyrite-proustite, boulangerite acanthite, and native silver. The Hazen mineralogy indicated that silver was also in solid solution with pyrite and possible sphalerite. Most of the silver-bearing minerals mentioned responded well to flotation when properly liberated. In a sequential lead-zinc flotation circuit, most of the silver minerals should report to the lead concentrate.

Photomicrographs and back-scatter images from a Scanning Electron Microscope (SEM) were taken by Hazen Research and confirmed this complex, fine-grained mineralogy. However, coarse galena grains were also noted in some samples. An example showing both complex and coarse liberated particles from Hazen Research on the Dawson composites is shown in Figure 13-5. Friebergite (f) is intergrown with pyrite – marcasite (yellow) and coarse, liberated galena (g) cleavage fragments.

Figure 13-5: Dawson Composite D, minus 500µm sinks (Hazen Research)



13.4 Grindability Tests

A summary of the grindability results is provided in Table 13-4. The number of samples tested per zone is also provided. The following tests were performed on Corani samples:

- Low Energy Impact Test, LEIT
- SAG Mill Comminution, SMC
- SAG Power Index, SPI
- Bond Rod Mill Work Index, BRWI
- Bond Ball Mill Work Index, BBWI
- Abrasion Index
- Point Load Test, PLT
- Unconfined Compressive Strength, UCS

Table 13-4: Comminution Test Results Summary

Parameter	LEIT Kw-h /mt	Dwi kWh/m^3	SMC Axb	SPI min	Crusher Index	BRWI kWh/t	BBWI 75 kWh/t	BBWI 106 kWh/t	Abrasion Index	PLT Is 50 Mpa	UCS Mpa
No. Tests	26	26	26	36	36	26	10	26	28	17	13
No. Este	13	13	13	15	15	13	2	13	13.5	8	6
No. Minas	9	9	9	15	15	9	6	9	10	7	7
No. Main	4	4	4	6	6	4	2	4	4.5	2	-
Average	6.30	2.35	111	35.6	26.9	10.2	15.5	14.6	0.17	1.47	31.8
Max	10.2	3.98	210	90.6	49.8	13.4	19.2	18.6	0.49	2.71	52.7
Min	2.68	1.17	60.8	15.3	14.1	6.53	12.3	10.1	0.03	0.67	23.5

Note: The number after BBWI refers to the closing screen size used for the test, in µm.

Based on these results, the ore appears to be of medium hardness with respect to SAG and ball milling.

13.5 Sequential Flotation Tests

Both bulk and sequential flowsheets were evaluated in several test programs. Much of the flotation testing, conducted by SGS, focused on the production of a marketable lead concentrate using the sequential flotation flowsheet. The separation of clean marketable lead and / or zinc concentrates was successful on only a portion of samples. For some samples, zinc or pyrite was recovered uncontrollably to the lead concentrate resulting in lower grade products. For other samples, flotation response was limited, leading to very low recoveries.

G&T conducted sequential flotation tests on a small sub-set of samples with favorable flotation response and suitable head grades.

Due to the improved marketability of the separate lead and / or zinc concentrates over a bulk concentrate; a sequential flowsheet was used for the design of the plant. The following subsections refer to the development of that sequential flowsheet.

13.5.1 Effect of Particle Size

The effect of primary grind was not studied in great detail for many of the test programs. Much of the testing was devoted to various depression and collector regimes used to control sphalerite (zinc) flotation in the lead circuit. However, there were series of tests conducted in the SGS 2008 and 2009 campaigns that examined the effect of primary grind size.

Rougher test data for the 2008 tests was taken from composites A and B, which represented only the FBS geological classification. The test data for the 2009 test program was from the Ag-Pb composite, which was a mix of geological classifications.

The effect of primary grind was more pronounced for silver when compared to lead as shown in Figure 13-6 and Figure 13-7. The expected trend of better metallurgical performance was often observed for the finer grind sizes early during the flotation stages, equating to the low rougher mass recovery. As rougher mass recovery was increased, the effect of grind size was minimal. The data would suggest that if sufficient rougher mass recovery is achieved, primary grind sizes of 100 micron K80 or coarser could be used. There may be potential to further coarsen primary grind size.

Figure 13-6: Relationship between Grind Size and Silver Rougher Recovery

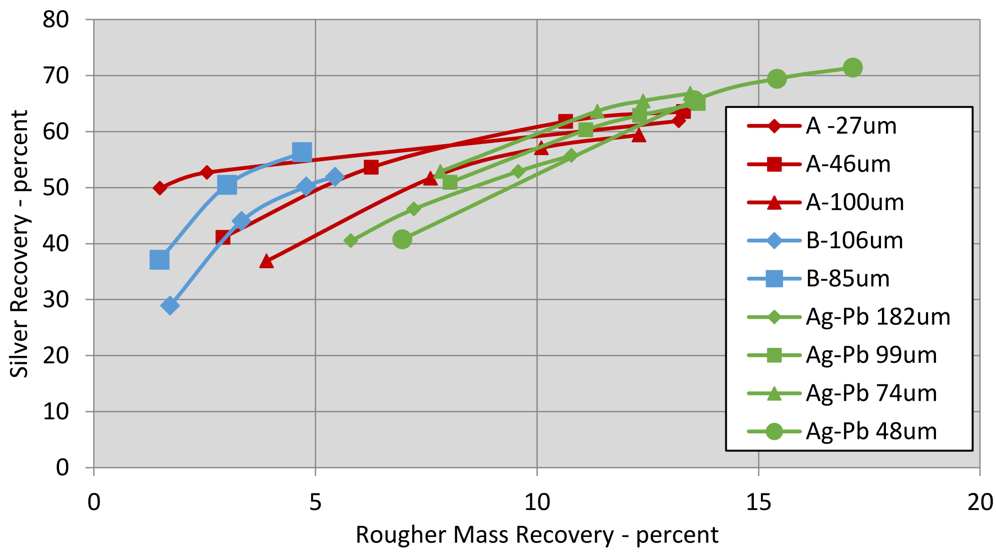
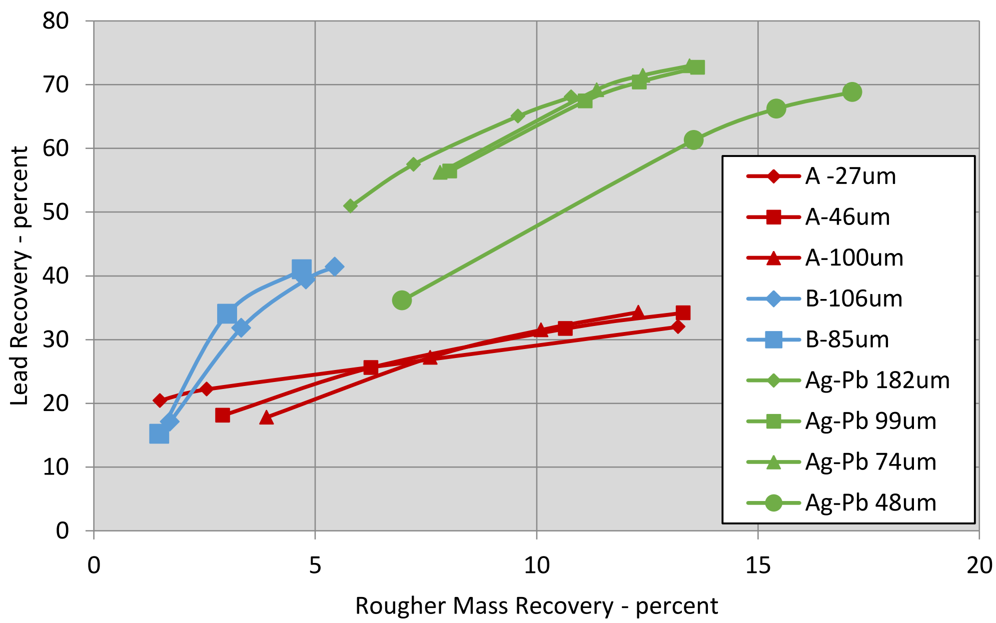


Figure 13-7: Relationship between Grind Size and Lead Rougher Recovery



13.5.2 Effect of Departments

A challenge with many of the Corani deposit samples was the poor selectivity between lead and zinc in the lead flotation circuit. The majority of the testing conducted was devoted to improving the selectivity of the lead flotation circuit against the flotation of sphalerite (zinc).

To tackle the activation of sphalerite, several depression schemes were employed. The depression schemes could be classified into general categories as follows:

- Zinc sulphate ($ZnSO_4$) – used in combination with selective collectors
- Zinc sulphate/cyanide ($NaCN$) or Zinc oxide/cyanide (ZnO / $NaCN$) – most common depressant scheme for zinc depression during lead flotation. Very effective to combat copper ion activation of sphalerite and pyrite.

- Sulfite/Sulfide includes reagents Na2SO3, Na2S, SO2 – less common, but effective to sphalerite, pyrite depression when cyanide is not permissible. High dosages of these reagents will also depress galena (lead).

The development of depressants used for most of the locked cycle tests occurred while testing of composites A, B, B2, and G during the SGS program in 2008-2009. The most complete matrix of testing was completed on sample G. Results of zinc depression are displayed in Figure 13-8 an Figure 13-9. Unfortunately, there were no comparable tests without depressant.

Figure 13-8: Effect of Depressants on Lead Rougher Flotation – G Composite (FBS)

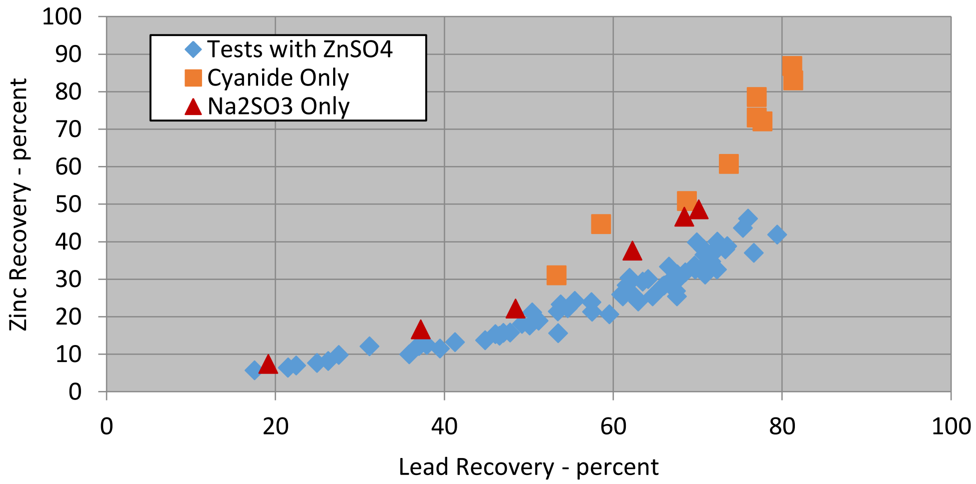
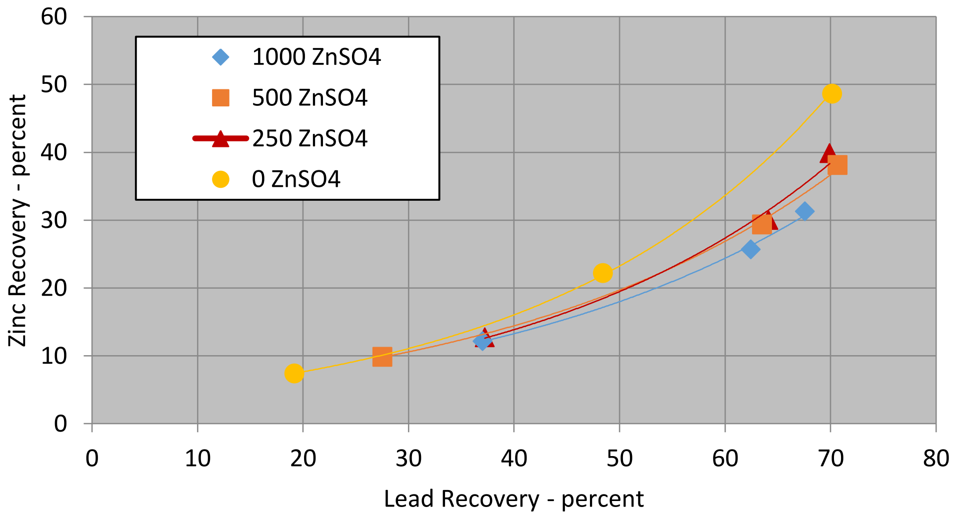


Figure 13-9: The Effect of Zinc Sulphate Dosage on Zinc Depression from the Lead Rougher Flotation, Composite G (FBS)



As shown in the figures, the best selectivity was achieved when zinc sulphate ($ZNSO_4$) was added. When cyanide was added without zinc sulphate, the activity of zinc was actually increased. It should be noted that one test was conducted with both zinc sulphate and cyanide and the results mirrored the zinc sulphate only conditions. Sodium sulphite (Na_2SO_3) alone did not appear to be effective for suppressing zinc in this sample.

To further investigate the effects of zinc sulphate, there were a number of tests that utilized variable dosages of zinc sulphate with the other variables remaining constant. The results of these tests are displayed in Figure 13-9.

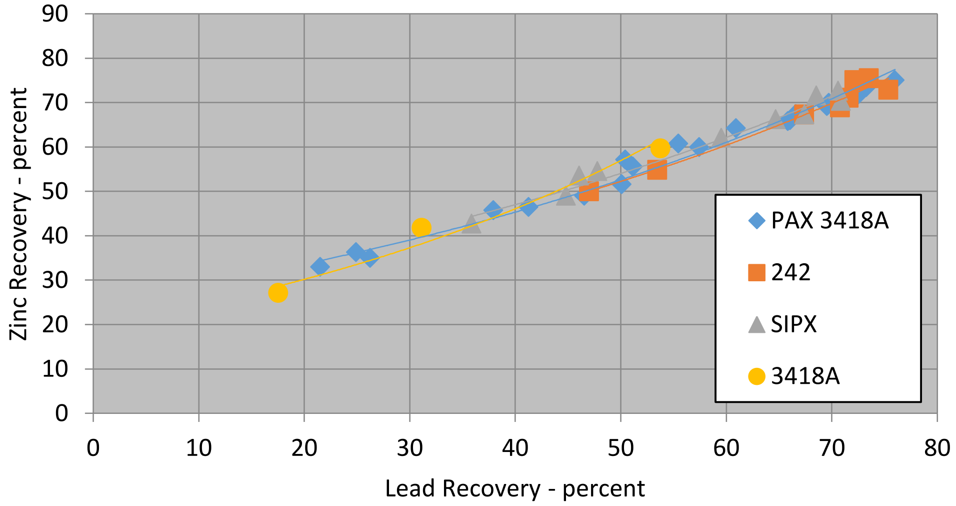
The curve indicated that the addition of zinc sulphate improved lead flotation selectivity against sphalerite. There was a slight improvement with increased dosage. As the Project continued, a depression regime was established using zinc sulphate, cyanide and sodium sulphite. The data generated on composite G was not verified on any other composite; however, the inclusion of zinc sulphate in the depression scheme may have been the most important aspect of controlling zinc in lead flotation in all of the following tests. There may be an opportunity to reduce or eliminate the use of sodium sulphite and cyanide.

13.5.3 Collectors

Again, much of the focus of flowsheet development program was dedicated to developing selectivity between lead and zinc in the lead flotation circuit. Several collectors were used, namely, Cytec products 3418A, 242, and 404. Xanthates were also included in combination with Cytec collectors. Potassium xanthate (strong) and sodium isopropyl xanthate (weaker) were primary xanthate collectors used. Xanthates were also used in the zinc circuit.

Unique to this Project, a theoretical application of emulsified diesel oil (EDO) was also considered. EDO was added after an initial low dosage application of a lead selective collector. The EDO acts as an extender, enhancing the hydrophobicity of the selective collector. This collector extender was widely used for much of the testing. Composite G had several combinations of collector tested and isolated (all other conditions constant). Figure 13-10 displays the selectivity differential between the different collectors.

Figure 13-10: The Effect of Collector Type on Lead Rougher Flotation Selectivity against Zinc, Composite G (FBS)

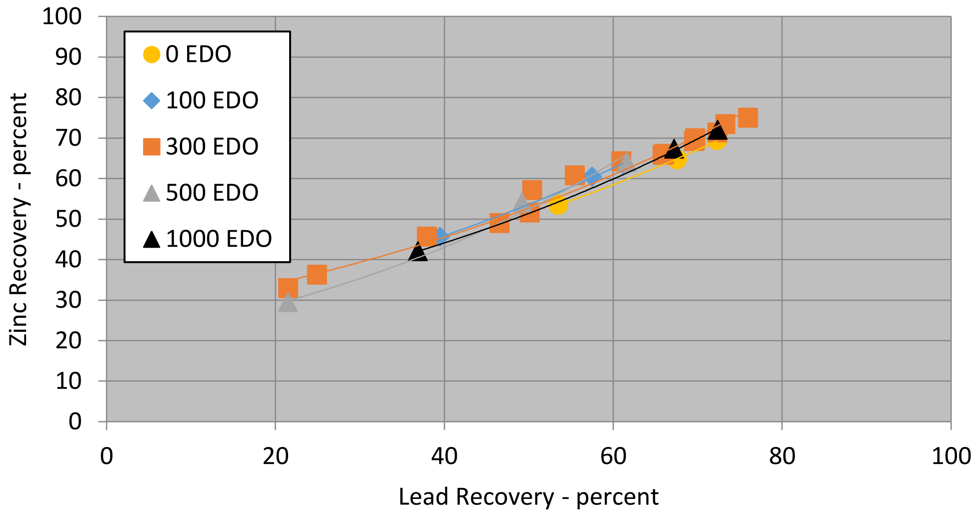


As shown, the use of different collectors did not have a significant impact on selectivity between lead and zinc in the lead rougher flotation. The dosage of collector did shift the results up the curve (higher dosages) or down the curve (lower dosages)

Using the same series of rougher tests, the effects of EDO were isolated. EDO did not impart any gains in selectivity, as shown in Figure 13-11. This conclusion was reached later in testing at SGS in 2011.

Based on these results, it would appear that the use of lower cost xanthates should be considered. As with all collectors, the dosage rate is an important factor to controlling selectivity in the lead circuit.

Figure 13-11: The Effect of EDO on Lead Rougher Flotation Selectivity against Zinc, Composite G (FBS)



13.5.4 Activators

In the lead circuit, activators such as sodium hydrosulfide and hydroximate collectors were briefly tested as a means of activating and recovering non-sulphide lead. The test results were discouraging, indicating that the plumbogummite, the dominate non-sulphide lead mineral, did not respond to these reagents.

In the zinc circuit, activation of zinc was accomplished with copper sulphate ($CuSO_4$). This is standard practice in industry. Dosages of copper sulphate required ranged between 100 and 350 g/tonne.

13.5.5 pH Regulators

The use of pH regulators can be effective in controlling pyrite when used in combination with selective collectors and other depressants. In the lead circuit, controlling the pH to 8 to 9 is common to help depress pyrite and other iron sulphides. Above pH 9, lead depression occurs. The natural pH of many of the samples tested was well below 7; most samples averaged 5.5. Increasing the pH with lime in the lead circuit was ineffective and often resulted in increased activity of sphalerite flotation. Decreasing the pH with sulphuric acid was also investigated but showed no advantage.

In 2011, a global composite (Ag-Pb-Zn) used for detailed flowsheet development indicated issues with pyrite activation as well as zinc activation in the lead flotation circuit. To combat issues of pyrite activation, sodium carbonate ($NaCO_3$) was used effectively to reject pyrite and increase the grade of lead in the concentrate for this composite. Many other composites did not require the use of sodium carbonate to produce high-grade lead concentrates.

For the zinc flotation circuit, the addition of lime to pH 11 or 11.5 was used for most of the composites tested.

13.5.6 Surface Modifiers

There were many surface modifiers tested, mostly to improve the quality of the lead concentrates. These included sodium silicate (silicate mineral depressant and dispersant) and starches (CMC and guar gum for silicate depressants). None of these reagents were useful in improving the metallurgical response of the flotation circuits.

13.5.7 Rougher Concentrate Regrind

Due to the primary focus on chemistry, very little coordinated testing was devoted to investigating the effects of regrind. The quantitative mineralogy suggested that some of the mineralization (FBS) would require very fine primary and rougher regrind targets to achieve acceptable concentrate grades. Most of the testing conducted in 2008-2009 by SGS did only semi-quantitative sizing analysis. The target regrind size for both the lead and zinc regrinds appeared to be between 20 and 30 micron K80 in these programs.

Based on the grain size data from the mineralogy, this target for the majority of the mineralization would be reasonable. The mineralization identified as CSC could utilize a coarser regrind and primary grind size target.

13.5.8 Summary of Sequential Locked-Cycle Test Results

While many of the locked-cycle tests had different conditions applied, the average for all tests performed is displayed in Table 13-5.

Table 13-5: Summary of Sequential Locked-Cycle Test Data

Zone	No. Samples	Pb Con Gr - % or g/t		Rec to Pb Con - %		Zn Con Gr - % or g/t		Rec to Zn Con - %	
		Pb	Ag	Pb	Ag	Zn	Ag	Zn	Ag
CSC	2	68	1620	79	70	52	729	72	12
FBS	17	50	2569	50	49	52	462	77	23
PM	6	50	1269	62	46	56	251	59	12
QSB	2	47	3386	64	64	-	-	-	-
FeOx	1	waste							
MnO	2	waste							

Note: Ag grades are denoted in g/tonne, all other assays are in percent.

No samples representing the PG (plumbogummite) zone only were tested in the 2007-2008 SGS programs. Such a sample would need to be obtained and tested to generate metallurgical performance predictions. However, given the poor metallurgical response of PG samples to a bulk flowsheet, it is unlikely such mineralization would respond well to a sequential flowsheet.

13.5.9 Concentrate Quality

A suite of the lead and zinc concentrates produced by SGS from locked-cycle testing was assayed for a series of minor elements. Results are summarized for the lead and zinc concentrates, respectively, in Table 13-6 and Table 13-7.

Table 13-6: Minor Elements in Lead Concentrates

Element	Units	Lead Concentrate from Locked Cycle Test on Composite											
		U	M	G	R	Main 1	3 zone AgPbZn Ave	1-5 yr	Q	T	Minas 1	Minas 3	Average
Ag	g/t	2448	1776	1678	465	810	1576	2100	1752	3696	1493	2173	**1815**
Pb	%	55.5	55.5	51.2	58.9	53.6	54.8	51.0	62.6	51.1	73.0	81.1	**58.9**
Zn	%	7.18	4.11	9.03	2.60	6.10	8.15	8.10	0.98	5.40	0.82	2.90	5.03
Cu	%	4.50	1.70	1.90	0.38	1.90	1.25	1.62	0.52	2.00	1.50	0.35	1.60
Au	g/t	-	-	-	-	-	0.29	0.37	-	-	-	-	0.33
S	%	-	-	-	-	-	20.3	21.3	-	-	-	-	20.8
C(t)	%	-	-	-	-	-	0.27	1.77	-	-	-	-	1.02
Cl	g/t	-	-	-	-	-	15	<10	-	-	-	-	<10
F	%	-	-	-	-	-	0.020	0.014	-	-	-	-	0.013
Hg	g/t	6.2	4.4	23.9	4.9	30.2	22.1	16.8	47.0	15.6	10.0	1.6	16.6
Al2O3	g/t	7366	4156	8122	14545	4911	3950	0.61	11333	3400	8878	1889	6232
As	g/t	510	1500	3000	2500	440	525	2750	740	910	470	<40	**1215**
Ba	g/t	2800	360	360	34	200	225	396	23	2800	3400	190	981

Element	Units	Lead Concentrate from Locked Cycle Test on Composite											
		U	M	G	R	Main 1	3 zone AgPbZn Ave	1-5 yr	Q	T	Minas 1	Minas 3	Average
Bi	g/t	<200	<30	<30	<20	<20	62	74	<20	130	120	<20	<200
Ca	g/t	190	420	400	200	<40	240	232	2100	480	540	240	460
Cd	g/t	450	620	2800	2000	310	720	1390	5000	450	46	190	**1271**
Fe	%	5.6	11.0	8.6	7.6	10	9.45	8.1	5.1	13.0	3.0	1.3	7.5
Mg	g/t	600	190	79	190	78	115	117	220	940	480	150	287
Mn	g/t	370	410	160	130	320	540	278	170	1000	1400	240	456
Na	g/t	23	<30	<30	280	72	58.5	72	170	62	150	130	95
P	g/t	370	360	570	<150	<200	<200	266	<150	890	1600	<200	446
Sb	%	1.70	0.62	1.90	0.38	0.70	0.58	1.21	0.20	0.90	0.48	0.34	**0.82**
Se	g/t	<30	<30	30	30	30	30	30	30	<40	30	<30	<30
Sn	g/t	<25	<30	<30	<20	<40	<30	<20	110	<50	<20	<20	<110
SiO2	%	-	-	-	-	-	2.95	4.15	-	-	-	-	3.55

Note: The 3 zone mixed sulphide (Ag-Pb-Zn) Composite ("3 zone AgPbZn Ave" in the table above) result is an average of two suites of assays from two locked cycle tests conducted on this composite.

Table 13-7: Minor Elements in Zinc Concentrates

Element	Units	Zinc Concentrate from Locked Cycle Test on Composite							
		U	M	G	R	Main 1	3 zone AgPbZn Ave	1-5 yr	Average
Ag	g/t	286	272	371	288	250	385	463	**331**
Pb	%	1.27	5.76	4.86	6.63	2.16	4.04	4.80	4.22
Zn	%	56.4	55.8	53.0	49.3	58.0	51.9	52.3	**53.8**
Cu	%	0.50	0.30	0.37	0.52	0.28	0.29	0.63	0.41
Au	g/t	-	-	-	-	-	0.10	0.39	0.25
S	%	-	-	-	-	-	28.6	31.7	30.1
C(t)	%	-	-	-	-	-	0.20	0.16	0.18
Cl	g/t	-	-	-	-	-	62.5	<10	34
F	%	-	-	-	-	-	0.021	0.008	0.014
Hg	g/t	23.3	38.8	67.3	47.0	55.1	89.1	69.4	**55.7**
Al2O3	g/t	4700	2078	4345	11334	8122	13750	2500	6690
As	g/t	600	890	170	740	230	285	722	520
Ba	g/t	970	140	200	23	390	465	179	338
Bi	g/t	<200	<30	<30	<20	<20	<50	<20	<50
Ca	g/t	1400	800	920	2100	1100	1835	1090	1321
Cd	g/t	1800	3300	5200	5000	1600	2650	4940	**3499**
Fe	%	4.6	3.8	3.1	5.1	5.3	4.6	5.2	4.5
Mg	g/t	960	1100	70	220	140	305	71	409
Mn	g/t	540	370	130	170	190	555	287	320
Na	g/t	<10	<30	<30	170	100	190	49	78
P	g/t	240	230	370	<150	<200	370	<200	212
Sb	%	0.36	0.08	0.16	0.20	0.36	0.145	0.21	**0.22**
Se	g/t	<30	<30	<30	<30	<30	<30	<30	<30
Sn	g/t	62	<30	110	110	<40	<30	57	56
SiO2	%	-	-	-	-	-	7.65	2.66	5.16

Note: The 3 zone mixed sulphide (Ag-Pb-Zn) Composite ("3 zone AgPbZn Ave" in the table above) result is an average of two suites of assays from two locked cycle tests conducted on this composite.

The lead concentrates that SGS assayed, on average, graded 59 percent lead and 1,815 g/tonne silver. The lead grade, particularly, is on the higher end of what is achievable for Corani mineralization. The concentrates, on average, graded about 0.8 percent antimony, which would be expected to result in smelting penalties. Arsenic, mercury, and cadmium were also elevated, and there may also be penalties applied for these elements.

The zinc concentrates assayed by SGS, on average, graded 53.8 percent zinc and 331 g/tonne silver. Mercury was elevated in these concentrates, grading on average 56 g/tonne in the zinc concentrates. At this level, penalties, if not marketability issues, would be anticipated. Antimony and cadmium were also elevated in these concentrates, and potential penalties may be applicable.

13.6 Alternative Flowsheet Arrangements

Bulk flotation testing was conducted for various samples in the Dawson, G&T, and SGS programs. For the SGS program, the sample set that was tested for bulk flotation may be biased towards samples with poor flotation response. But Dawson and G&T both seem to test a full range of samples, and some conclusions as to the variability in metallurgical response to flotation can be drawn from that work.

In the 2006 test program, conducted by Dawson Metallurgical Laboratories, Ltd., a suite of 12 samples was tested for cyanidation leaching response. This sample selection appears to be unbiased, whereas, in subsequent programs, only a selection of samples was subjected to cyanidation bottle roll tests, presumably due to poor flotation response. As such, only limited information can be drawn from those datasets. Dawson also conducted testing on two samples evaluating a combination of flotation and cyanidation.

The following sub-sections discuss the result of tests conducted using these alternative flowsheets.

13.6.1 Bulk Flotation

In the program of testing by G&T, a test was conducted on each of 71 samples using the same reagent regime. In these tests, the objective was to recover all the sulphides into a bulk concentrate. A nominal primary grind sizing of 75 µm K80 was used. Only collectors, SIPX, and 3418A were added to the bulk roughing stage, with lime also added in the regrinding stage. Since no depressants were added beyond the elevated pH in the cleaner, higher recoveries but poorer concentrate grades were obtained as compared to the sequential flowsheet.

Upon analysis of the G&T results, it is clear that the different ore types designated by BCM were useful to delineate metallurgical performance based on these results. A summary of average batch cleaner test performance, obtained from each ore-type, is provided in Table 13-8.

Table 13-8: Average Bulk Circuit Performance by Ore-type (based on G&T Data)

Ore Type	No. Samples	Ave Head Grade - % or g/t				Con Grade - % or g/t			Recovery - %		
		Pb	Zn	Ag	S	Pb	Zn	Ag	Pb	Zn	Ag
CSC	6	3.70	0.94	137	3.55	27.7	7.3	1110	82	68	78
FBS	26	1.83	1.59	96	3.32	15.8	12.5	1200	64	57	71
PM	15	1.37	2.32	62	5.87	10.4	11.8	444	76	49	75
QSB	7	1.02	0.18	75	1.67	13.9	0.9	12723	11	3	46
FeOx	4	0.52	0.09	71	1.20	9.8	2.9	2665	34	36	55
MnO	6	1.75	0.23	64	3.04	10.6	1.5	3564	5	5	31
PG	7	1.23	0.11	27	0.69	9.9	2.1	1277	5	10	25

Note: Ag grades are denoted in g/tonne, all other assays are in percent.

13.6.2 Whole Ore Cyanidation Tests

Dawson utilized a nominal primary grind sizing of 75 µm K80 to conduct the cyanidation bottle roll tests. The 96-hour silver extraction and cyanide consumption data for the tests conducted by Dawson are summarized in Table 13-9.

Table 13-9: Dawson Cyanidation Leaching Results

Composite	Ag Extraction 96 hours - %	Ag Grade – g/t		NaCN kg/t
		Residue	Calculated Head	
1	56.9	99.0	229.0	4.5
2	58.8	110.0	266.0	4.6
3	54.4	105.0	230.0	3.4
4	70.5	41.0	139.0	3.0
5	64.6	53.0	150.0	4.6

Composite	Ag Extraction 96 hours - %	Ag Grade – g/t		NaCN kg/t
		Residue	Calculated Head	
6	79.1	30.0	143.0	5.7
A	48.8	63.0	122.0	4.0
B	39.9	64.0	100.0	7.4
C	56.9	36.0	93.0	4.9
D	50.1	60.0	110.0	4.3
E	45.1	52.0	94.0	2.8
F	25.5	140.0	188.0	2.9
Wt. Ave.	**54.1**			

On average, just over half the silver was extracted using cyanidation bottle roll tests. Silver extraction kinetics tended to be slow after 8 hours.

Cyanidation testing was only conducted on selected samples, generally of poor flotation response, in the G&T and SGS testing programs. However, results showed that under conditions almost the same as used by Dawson, similar silver extractions were obtained by G&T and SGS on these samples. Using a weighted average, about 51 percent of the silver was extracted from the samples tested by G&T and SGS.

Average performance for each deposit and ore type is summarized in Table 13-10. Results show that samples that have poor bulk flotation response (FeOx, QSB, MnO, and PG) tended to respond more favorably to cyanidation. It may be possible to take advantage of this by pursuing a flowsheet involving cyanidation of flotation tailings.

Table 13-10: Average Cyanidation Extraction by Deposit and Classification (G&T Data)

Zone	No. Samples	Wt. Ave. Ag Extraction - %
Deposit		
Este	9	38
Main	11	73
Minas	12	54
Ore Type		
FBS	8	35
PM	3	26
FeOx	3	70
QSB	5	81
MnO	6	59
PG	7	79

Note: Silver extraction averages were weighted using silver head grades.

13.6.3 Cyanidation Leaching of Flotation Tailings

The opportunity to process Corani mineralization using a combination of flotation and cyanidation was only evaluated by Dawson Metallurgical Laboratories in 2006. In that program, Dawson tested only two samples using this methodology. The silver recovery was superior to that obtained by the use of either flotation or cyanidation alone. Further testing would be required to determine if this flowsheet option has merit across a broad range of samples.

13.7 Previous Metallurgical Recovery Model

Discrete mineral domains were used to assign metallurgical performance in the previous feasibility study (M3, 2011). Previously, nine mineral types were identified for the Corani deposit. These mineral types were grouped into four metallurgical categories, and each category was assigned an average recovery or a grade-dependent recovery. These domains were defined by BCM geologists based on review of the drill core and development of cross-sections of the deposit using professional judgement. From the cross-sections, solids of the domains were created and used to assign metallurgical parameters to the block model. While the current geometallurgical approach supercedes this methodology for predicting metal recoveries; it is important to mention the previous work as the mineral domains are still relevant to the understanding of the Corani deposit mineral zonation and metallurgical responses.

Though these mineralization types generally indicated metallurgical performance, the behaviour for samples within each metallurgical type was highly variable, particularly within the mixed oxide/sulphide types (FBS).

Detailed analysis of the mineralogical test data suggested that these mineralogical domains did not fully capture the transitional nature of the Corani deposit. New analysis of the metallurgical, QEMSCAN, and geologic data (presented in Section 13.8) indicated that much of the Corani ore is transitional material that ranges from slightly to strongly oxidized. Since metallurgical recovery is strongly related to the degree of oxidation, it was necessary to develop a new geometallurgical model that could represent variations in oxidation throughout the ore body, which are highly variable and inconsistent both vertically and horizontally. A new geometallurgical model was developed that utilizes mineralogy and other factors to provide more accurate predictions of metallurgical recovery. Those factors include elevation, zinc grade, and logged geologic data, especially key minerals such as galena, pyrite, goethite, and manganese oxides. These parameters serve as indicators for the degree of oxidation that has occurred and allow for a continuous estimate of metallurgical parameters. The new model provides a more accurate representation of the deposit's metallurgical response, particularly for the transitional material because it has the ability to represent the gradational boundary between sulfide and oxide material indicated by mineralogical data.

13.8 Optimized and Current Continuous Predictive Metallurgical Model

Corani is a silver-lead-zinc deposit that exhibits a high degree of variability with respect to metallurgical test results. Most of the metallurgical testing was based on composites constructed by geological classification. Some of the classifications were suitable for categorizing metallurgical response, for example, composites of CSC (Coarse grained silica-sulphide-celadonite) consistently performed very well. However, one of the categories which represented nearly half of the resource tonnage (FBS) had large ranges in metallurgical performance.

GRE and Tom Shouldice (TS) conducted an evaluation of the geometallurgy for the Corani Project to address recommendations made by Blue Coast Metallurgy (Blue Coast Metallurgy, 2011) in conjunction with the test results described above regarding metal recovery projections. At the time of the 2011 Feasibility Study, four geometallurgical types had been identified, and average recoveries were assigned to these material types for reserve calculations and mine planning, as described above. These geometallurgical types represented varying degrees of oxidation and as a result, distinct differences in metallurgical behaviour were observed between types. However, within each assigned met type, large ranges in recovery were observed. To further refine metallurgical recovery projections, the available metallurgical, mineralogical, geological, and spatial data were extensively reviewed.

A statistical analysis was completed by GRE indicating that several measurable geological parameters could be used to make metallurgical predictions.

A number of advanced statistical methods were used to better understand the drivers of metallurgical behaviour and identify data within the existing drill hole database that could be used to predict recovery for the deposit. These methods included cluster analysis and recursive partitioning. Since the conceptual understanding of the Corani deposit relies on various types of mineralization, it is expected that characteristics that drive recovery within one mineralization may be different or have a different magnitude in another mineralization. This was previously recognized from a qualitative standpoint (i.e., the qualitative assignment of Met types I through IV). The advanced statistical methods allow for a quantitative assessment of this behaviour.

Using these methods, zinc grade, elevation, oxide minerals, pyrite, and the form of lead (galena vs. phosphate) were identified as the best predictors of lead recovery; zinc grade, elevation, copper grade, and pyrite were identified as the best predictors of zinc recovery; and lead recovery was identified as the best predictor of silver recovery in the lead concentrate. To model the complex relationship between these parameters and recovery, multivariate adaptive regression splines were used to develop a model capable of predicting recoveries based on this data. Multivariate adaptive regression splines are a form of multiple regressions that has the flexibility to model non-linear relationships between variables. The method also uses data partitioning, which allows the model to identify characteristics within the dataset that potentially lead to different outcomes.

Though this method appears complex, it combines the benefits of a model based on ore type characterization (Met Types I –Met Type IV) and a model using regression. It partitions data based on similarities (similar to ore type characterization) and develops a regression model for predicting a continuous recovery result (similar to regression modelling). The result is a single model equation that can be used to predict recovery across all ore types.

To select a robust model, subsets of the dataset were selected randomly and used to develop a calibrated model; the remaining data was used as a “blind” dataset to validate the model. This was done 1000 times, and the best performing model (based on calibration and validation results) was selected as the final model.

To create an overall balance for lead, silver, and zinc, the concentrates were assumed to be constant grade. The grades of the concentrates were determined by the averages of locked-cycle testing.

The sections below give a brief summary of each model used for recovery predictions. Additional details regarding the complete statistical analysis were reported by GRE in a separate document.

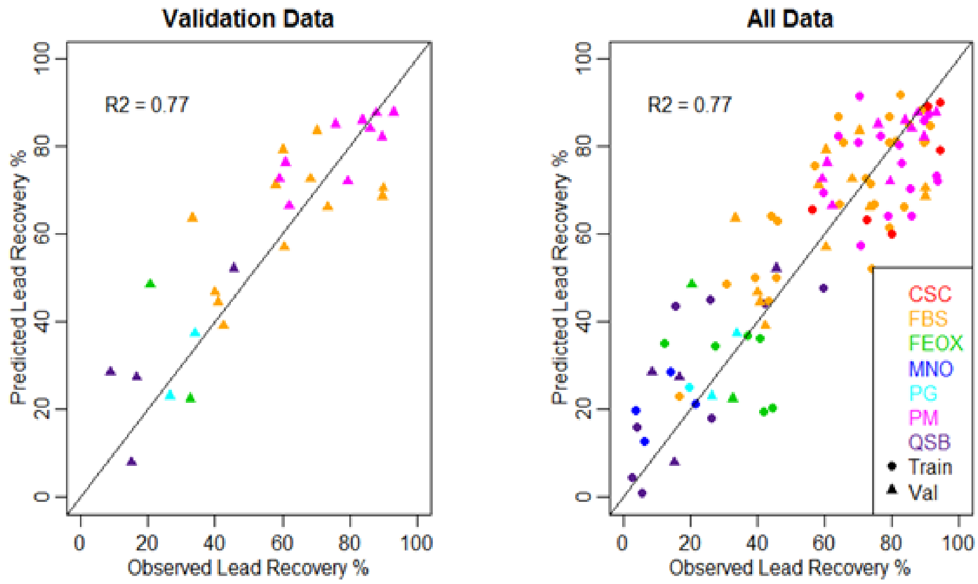
13.8.1 Recovery of Lead to Lead Concentrate

The model for predicting lead recovery to the lead rougher concentrate included the following variables:

- Zinc feed grade
- Mine elevation
- Geological logging estimates of galena, goethite, manganese oxide and pyrite.

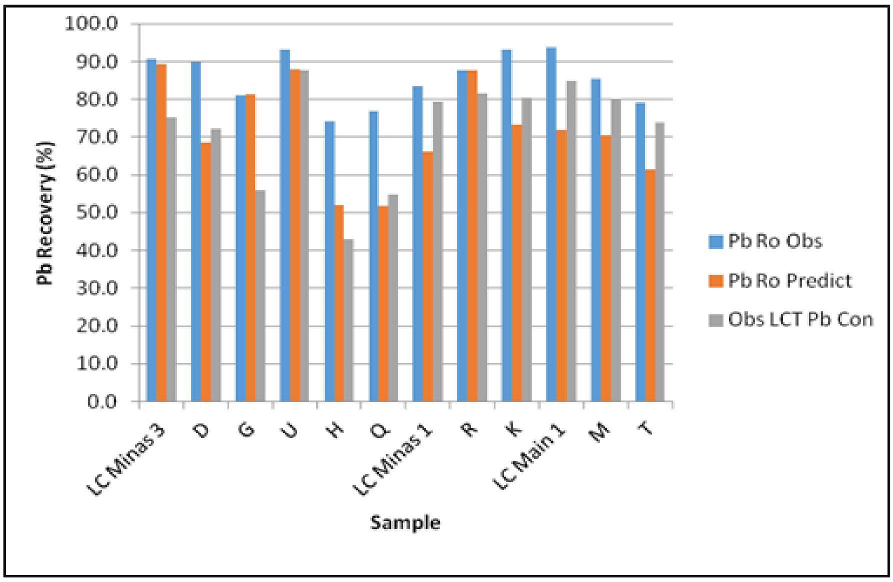
To ensure the robustness of the selected regression parameters, 70% of the data was randomly selected (the training dataset) and used to estimate the regression parameters; the remaining 30% (the validation dataset) was used to validate the model. This process was repeated 1000 times. The model with the most stable regression parameters and best calibration-validation performance was selected. The R2 of the training dataset for the selected model was 0.77, and the R2 of the validation dataset was also 0.77. The R2 considering all of the data (both validation and calibration data) was 0.77. The performance of the model for the validation dataset and across all data is shown in Figure 13-12. The colors in the figure represent the mineralization types assigned to samples using the prior geometallurgical approach.

Figure 13-12: Selected Model with Training Data and Validation Data



Predicted lead rougher recoveries were compared to the final lead concentrate recoveries observed during locked-cycle tests for the 12 samples in which both locked- cycle test and batch test results were available. The results indicate that the model developed using lead rougher results from batch testing is a good predictor of final lead concentrate recovery. A comparison of observed lead rougher recoveries from batch testing, predicted lead rougher recoveries from batch testing, and observed final lead concentrate recoveries is shown in Figure 13-13.

Figure 13-13: Comparison of Predicted and Observed Lead Recovery from LCT



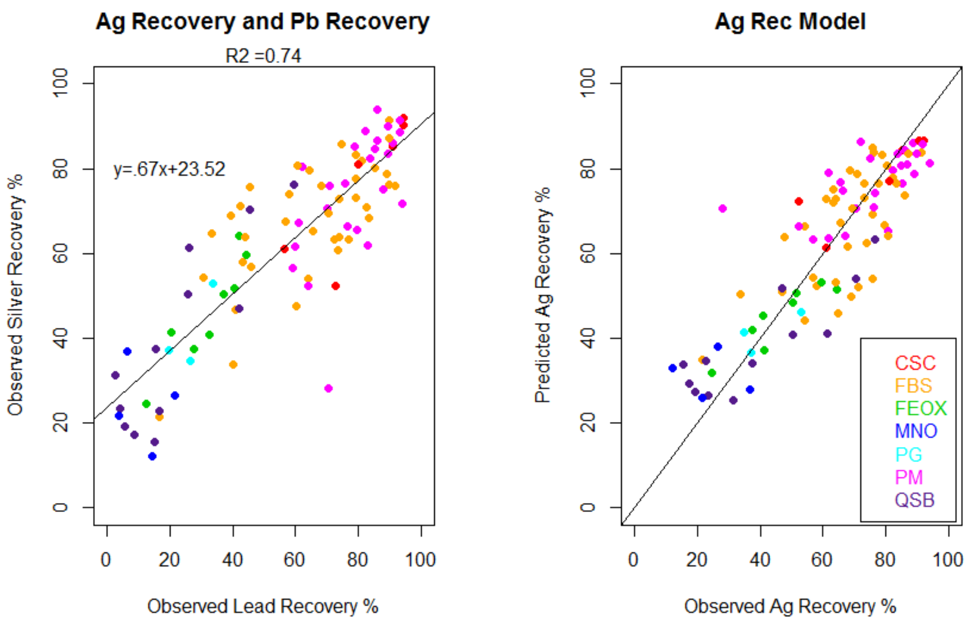
Based on the LCT analysis, the final lead recovery to the lead/silver concentrate was calculated using the following equation:

$$Pb\ Recovery\ to\ Final\ Lead/Silver\ Concentrate$$
$$= 61.9 - 40.9 * max(0{,}0.57 - zinc) + 7.7 * max(0, galena - 0.38) + 45.4 * max(0{,}0.37 - goethite) - 0.12 * max(0, elevation - 4891) + 32.9 * max(0{,}0.27 - MnOxi) - 6.21 * max(0, Pyrite - 1.07) - 16.4 * max(0{,}1.07 - Pyrite)$$

13.8.2 Silver Recovery

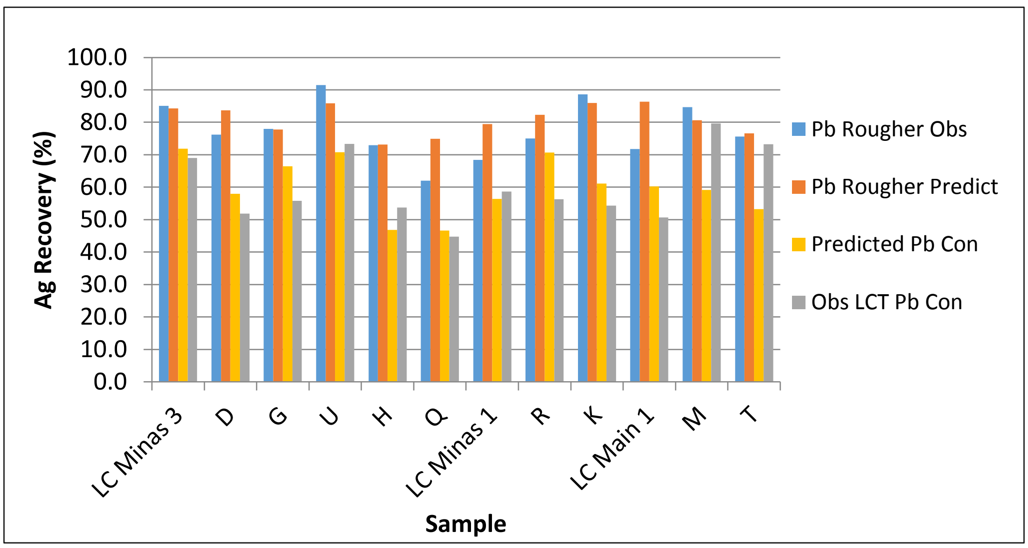
A cursory analysis of silver revealed that the strongest predictor of silver recovery was lead recovery. The relationship between silver and lead recovery to the lead rougher is shown in Figure 13-14. Though one major outlier does exist, due to the large number of samples, the equation is not strongly influenced by this observation.

Figure 13-14: Silver Recovery to Lead Rougher



The predicted silver recoveries were compared to LCT test results for samples where batch results and LCT results were available. Silver recovery to the lead rougher was estimated using predicted lead recovery to the lead rougher. The results show that the difference between the predicted silver recovery to the lead rougher and the observed silver recovery to the final silver/lead concentrate is on average 11.5%. This difference is assumed to represent the loss of silver during the lead cleaning stage. The final predicted silver recovery to the lead concentrate was estimated by subtracting 11.5% from the predicted silver recovery to the lead rougher. This yields an estimate of final silver recovery that closely matches the observed LCT test results (Figure 13-15).

Figure 13-15: Comparison of Predicted and Observed Silver Recovery to Lead Concentrate



Based on the LCT analysis, the final silver recovery to the lead/silver concentrate was calculated using the following equation:

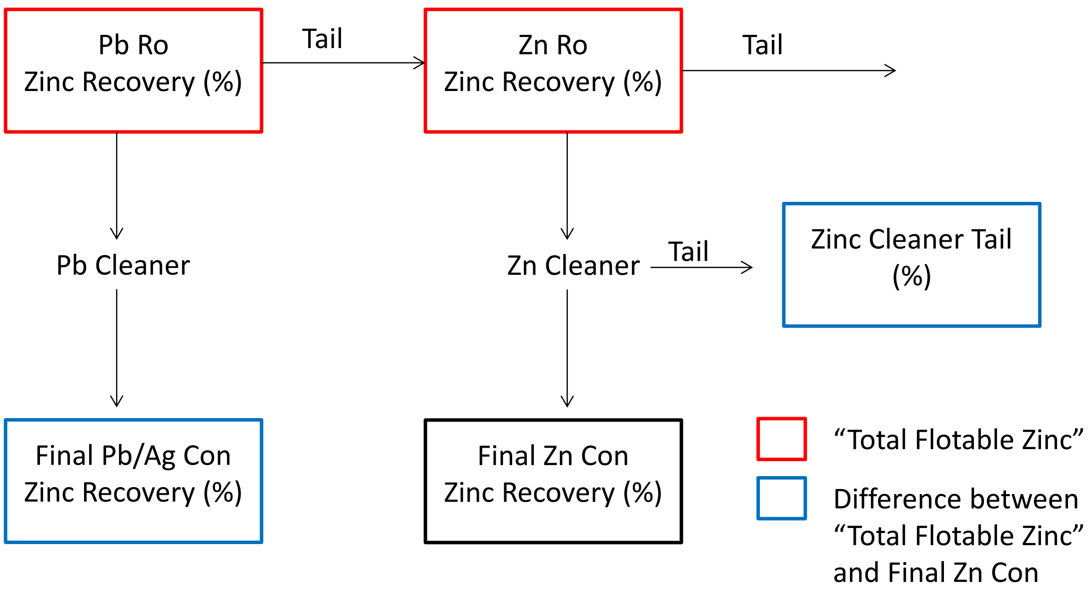
$$Ag\ Recovery\ to\ Final\ Lead/Silver\ Concentrate = 0.67 * Predicted\ Lead\ Recovery + 12$$

This is the same equation presented above (see Figure 13-14), for predicting silver recovery to the lead rougher, with the intercept adjusted to account for the loss of silver during the cleaning stage. This equation was used to assign final silver recoveries for the lead/silver concentrate to the Corani block model.

13.8.3 Zinc Recovery

Regression analysis for zinc was performed using “total floatable zinc,” which is the sum of the zinc that reports to the lead rougher and the zinc that reports to the zinc rougher during batch flotation testing. Non-floatable zinc, which is zinc that does not float in the lead rougher or zinc rougher, reports to the final zinc rougher tail. The difference between total floatable zinc and zinc recovered to the final zinc concentrate is the sum of the zinc that is recovered in the final lead/silver concentrate and the zinc removed during the zinc cleaning stage (cleaner tail). A schematic indicating the modelled recovery (“total floatable zinc”) is shown in Figure 13-16.

Figure 13-16: Schematic Indicating “Total Flotable Zinc” for Batch Tests



Based on the results from the above analysis, tests with lime additions to the lead rougher greater than 2,000 g/t were removed from the model dataset. From this test database, the test representing the best total zinc recovery result was selected for each sample. Also, as was done for the lead analysis, samples representing composites from multiple drill holes were removed from the potential training dataset. The resulting dataset included 58 of the original 72 samples considered for analysis.

To select the most robust model, a cross-validation process similar to the process employed for lead was used to develop the model for total floatable zinc. From the dataset, 70% of the samples were randomly selected to fit the model and the remaining 30% were used to validate the model. This process was completed 1000 times with 1000 different permutations of the dataset.

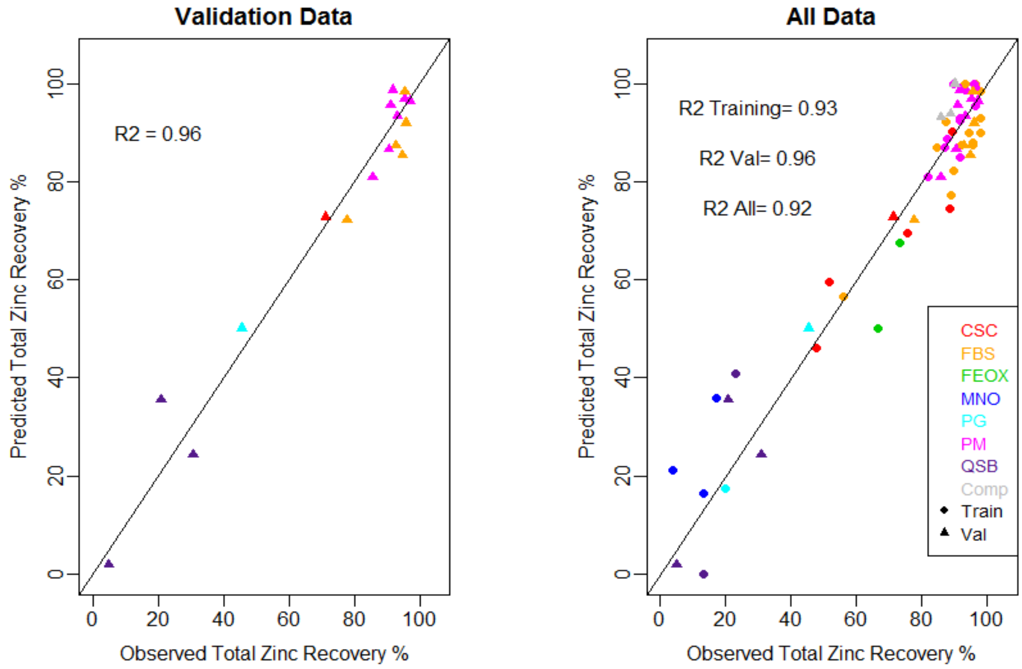
The independent variable combinations selected most frequently for the 1000 permutations were used for the final recovery equation. This combination had the highest validation and training set R^2, indicating the most robust model. The final total zinc rougher recovery model included zinc grade, elevation, copper, and pyrite in the following form:

$$Total\ Flotable\ Zinc = 93.9 - 50.6 * max(0{,}1.02 - zinc) - 0.15 * max(0, elevation - 4901) - 5.4 * max(0, pyrite - 1.9) - 11.2 * max(0{,}1.9 - Pyrite) + 104.1 * max(0, copper - 0.03) + 1620.2 * max(0{,}0.03 - copper)$$

The R^2 of the training dataset was 0.92, and the R^2 of the validation dataset was 0.96. The R^2 considering all of the data was 0.92. Once the model was limited to predicted zinc recoveries between 0 and 100%, the training R^2 improved to 0.93; the overall R2 remained the same.

The performance of the model for the validation dataset and across all data is shown in Figure 13-17.

Figure 13-17: Selected Model with Training Data and Validation Data

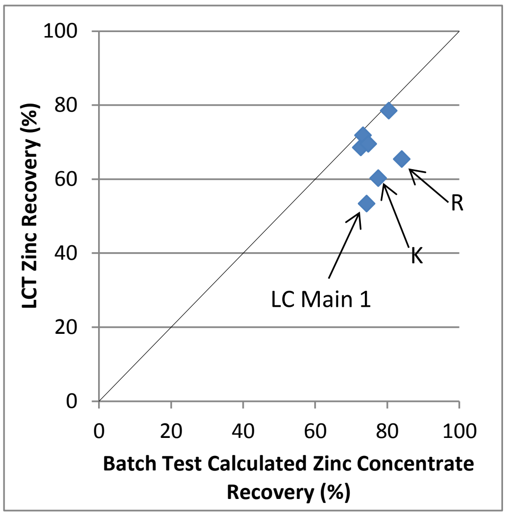


The results from the model were compared to the locked-cycle test results for the 8 samples where zinc recovery was reported. The average difference between the locked-cycle tests and the predicted total floatable zinc was 22.5%. This difference is anticipated to represent the zinc that reports to the final lead concentrate and the zinc that is lost during the zinc cleaning stage.

Three LCT test results were notable for lower-than-expected zinc recovery. The difference between the predicted total zinc recovery and the observed LCT recovery for samples K, R, and LC Main 1 were on average 34.5%, while the difference between the LCT zinc recovery and predicted total zinc recovery for the remaining samples was only 15.2%.

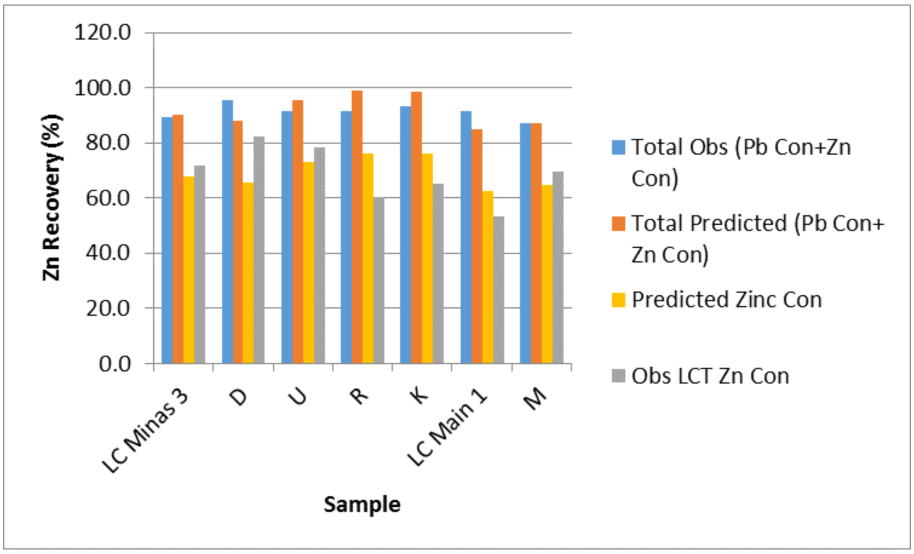
A comparison between the calculated final zinc concentrate recovery from batch tests is shown in Figure 13-18. The LCT result for these samples is much lower than what was achieved in batch testing. The remaining samples show a good match between the LCT result and batch testing. Considering this, the adjustment of 22.5% to the predicted total floatable zinc may be an over-estimate of the zinc lost to the lead rougher and the zinc lost to the zinc cleaner tail.

Figure 13-18: Comparison between Batch and LCT Results



A final zinc concentrate recovery was predicted based on the total predicted zinc recovery (Zn Ro+ Pb Ro) minus 15.2%, which is the average difference between the predicted recovery and the LCT zinc concentrate recovery result (not including samples R, K, and LC Main 1). The LCT zinc concentrate recovery, predicted total recovery, and predicted recovery to zinc concentrate are shown in Figure 13-19.

Figure 13-19: LCT/Predicted Zinc Recovery Comparison



13.8.4 Zinc Recovery to Final Zinc Concentrate

Based on the LCT analysis, the final zinc recovery to the zinc concentrate was calculated using the following equation:

$$\begin{aligned} &Zinc\ Recovery\ to\ Final\ Zinc\ Concentrate \\ &\quad = 78.7 - 50.6 * max(0{,}1.02 - zinc) - 0.15 * max(0, elevation - 4901) - 5.4 \\ &\quad * max(0, pyrite - 1.9) - 11.2 * max(0{,}1.9 - Pyrite) + 104.1 * max(0, copper - 0.03) \\ &\quad + 1620.2 * max(0{,}0.03 - copper) \end{aligned}$$

This is the same equation already presented for predicting total floatable zinc, with the intercept adjusted to account for the difference between total floatable zinc and final zinc recovery to the zinc concentrate (~15.2%). This equation was used to assign final zinc recoveries for the zinc concentrate to the Corani block model.

13.8.5 Zinc Displacement to Lead/Silver Concentrate

Based on the LCT results, the average displacement of zinc in the final lead/silver concentrate is 9%. Batch test results revealed that, in general, the lead cleaning stage was relatively effective in removing zinc from the final lead concentrate. Of the 97 batch tests used to create the lead rougher recovery model, 21 reported displacements for zinc in the lead concentrate following the cleaning stage. The average zinc displacement among these tests was 9.3%. With the exception of 2 outliers, most samples have zinc recoveries below 15%. Over 80% of the samples have zinc recoveries to the lead cleaner of less than 10%.

From these results, a recovery of zinc to the final lead concentrate was estimated to be roughly 9%.

13.8.6 Silver Recovery to Zinc Concentrate

Based on the tests used in the total floatable zinc model analysis, there does not appear to be a strong relationship between silver recovery to the zinc concentrate and zinc grades. Therefore, the zinc concentrate was assumed to contain an average silver grade. The grade was an average of the values determined in the locked-cycle testing. Silver recovery was then calculated once the zinc recovery and zinc concentrate mass was estimated from the zinc model.

13.8.7 Model Output Results

The models for recovery, along with the concentrate grade estimates, were input into the block modelling process to optimize the mining process. For the purposes of this study, the lead concentrate grade was assumed to be constant at 56.6% (average of all locked-cycle tests). Similarly the zinc concentrate grade was assumed to be 52.9% and contained 385 g/tonne silver (averages of all locked-cycle tests).

The block model simulation with the metallurgical models in place resulted in several pit shells at different revenue factors. Using the mine plan with revenue factor of 100, the estimates of metal recoveries were determined by mine operating year. These values are displayed for the lead concentrate and zinc concentrate in Table 13-11 and Table 13-12, respectively. Compared to the previous study (M3, 2011), the average lead recovery was reduced by 8% while the silver and zinc recovery both increased by 8%.

Table 13-11: Lead Concentrate Grades and Recoveries by Mine Schedule

Production Year	Tonnes t(dry) x 1000	Feed Grade (% or g/t)		Grade (g/t or %)		Recovery (%)	
		Ag	Pb	Ag	Pb	Ag	Pb
Year 1	5,328.75	95	1.14	4,560	50.0	65.6	59.8
Year 2	7,875	77	1.06	3,569	49.9	75.4	66.3
Year 3	7,875	93	1.24	4,370	53.8	67.7	62.4
year 4 to 5	15,750	62	1.10	3,086	53.8	68.5	67.3
Year 6 to 10	39,375	51	0.90	3,858	50.0	60.4	51.7
Year 11 to 18	62,868.9	35	0.76	2,354	50.2	64.2	64.5
LOM	139,072.6	50	0.90	3,010	51.0	64.3	61.1

Note: Ag grades are denoted in g/tonne, all other assays are in percent. Mine plan revenue factor of 100.

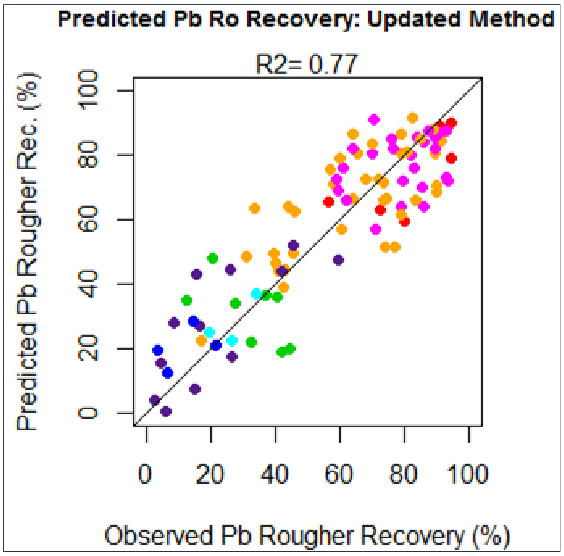
Table 13-12: Zinc Concentrate Grades and Recoveries by Mine Schedule

Production Year	Tonnes (000)	Feed Grade (%)	Grade (g/t or %)		Recovery (%)	
		Zn	Ag	Pb	Ag	Pb
Year 1	5,328.75	0.74	385	53.0	3.9	68.6
Year 2	7,875	0.98	385	53.6	6.6	73.0
Year 3	7,875	0.75	385	52.5	4.0	66.4
year 4 to 5	15,750	0.60	385	53.2	4.8	68.2
Year 6 to 10	39,375	0.52	385	53.2	5.9	55.7
Year 11 to 18	62,868.9	0.55	385	54.3	7.7	64.1
LOM	139,072.6	0.59	385	53.9	5.6	67.1

Note: Ag grades are denoted in g/tonne, all other assays are in percent. Mine plan revenue factor of 100.

The result of this work is a reliable, predictive geometallurgical model that has significantly improved predictions of metallurgical recovery which can be used in mine planning and grade control (Figure 13-20). The previous geometallurgical model relied on assigning average recovery values to large volumes of material based on ore type. Anticipated recoveries based on the new model closely fit recoveries observed during metallurgical testing. The recovery projections (based on the new model) for over half the samples tested is within +/-9% of the observed test recovery for lead, within +/-7% of the observed test recovery for silver, and within 5% of the observed test recovery for zinc.

Figure 13-20: Predicted vs. Observed Recoveries for New Predictive Geometallurgical Model



14 Mineral Resource Estimates

GRE estimated the Mineral Resources for the Corani Project in the first quarter of 2015. No new drilling or geologic information has been completed since the May 30, 2015 Optimized and Final Feasibility Study Technical Report was prepared. The resource model created in 2015 was used unchanged as the basis for the new Front End Engineering Design work and Technical Report. The following describe the estimation of Mineral Resources that was performed in 2015 and documented in the May 30, 2015 Technical Report.

A block model of the Corani deposit was developed as the basis for determination of the Mineral Reserves and Mineral Resources. This section summarizes the development of the block model as well as the development of the mineral resource. BCM's block model was updated by GRE to include drill hole data gathered since the 2011 model was created. The updated data also included new geologic cross-sections and updates to the mineral domains. Also, per previous authors' recommendations, the density model was updated with a statistical analysis of the relationship between metal grades and specific gravity. The drill hole database was updated with geologic logs and assays of primary recovery indicators: copper, goethite, manganese oxide, pyrite, and galena. These new grade labels were modeled along with the economically viable metals in the block model.

14.1 Block Model

The block model was developed using blocks sized 15 x 15 m in plan and 8 m high. The selection of the 8-m bench height was based on a study of dilution versus bench height and consideration of the mining equipment that might be used in the future for open pit production.

The model area is divided into three areas shown in Figure 14-1: Main, Minas, and Este. The model is large enough to contain all reasonable open pit configurations for the three resource areas. The total model size and block size are summarized in Table 14-1.

Figure 14-1: Resource Areas with Final Pit Configuration

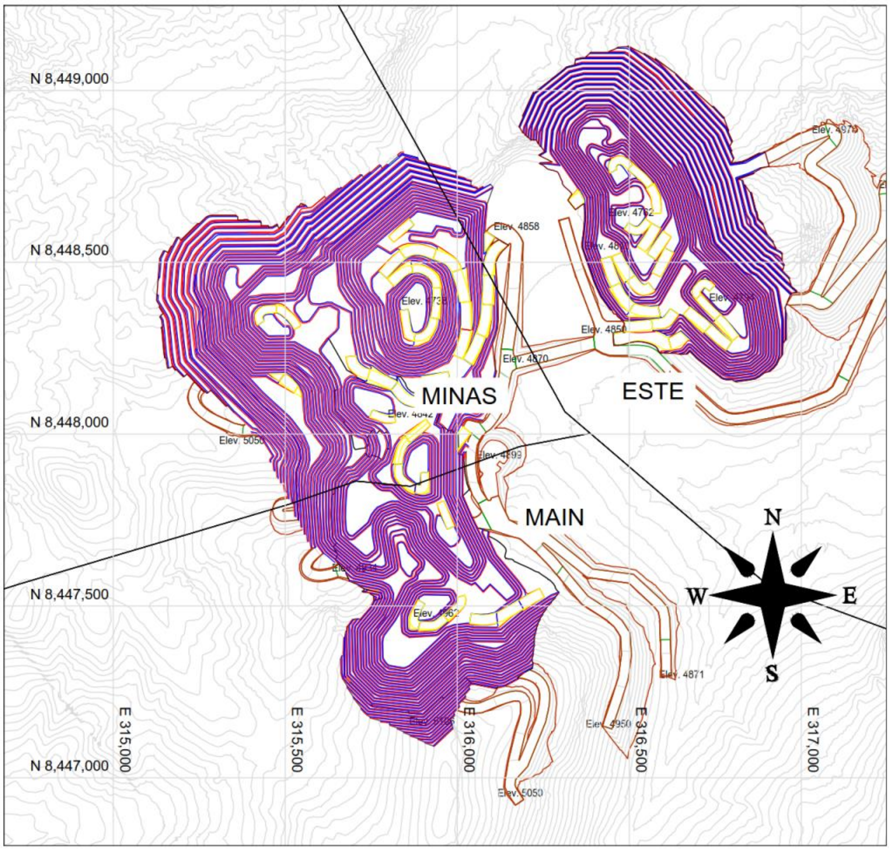


Table 14-1: Block Model Information

Parameter	X	Y	Z
Block Size (m)	15	15	8
Number of Blocks (m)	184	214	64
Model Limits	314,745	8,446,250	4,618
	317,505	8,449,460	5,130

The model was assembled in the Universal Transverse Mercator (UTM) coordinate system and is parallel to the UTM grid. Topographic information was assigned to the model based on topographic maps provided by BCM. The topography maps were consistent with field observations and the elevations of the drill holes at their coordinates.

14.1.1 Rock Type Boundaries

The Corani deposit lithology is comprised of Tertiary pre-mineral tuff and post-mineral tuff overlying Paleozoic sedimentary units.

The mineralized volcanic tuff was emplaced prior to mineralizing events (pre-mineral), and contains the economic minerals of interest. Post-mineral volcanic tuff is barren and was emplaced subsequent to the mineralizing events. The mineral reserves and resources are completely contained within the pre-mineral tuff.

The basement sediments at Corani are generally barren, although there are local occurrences of mineralization within those units. Potential exists for additional mineralization to be found in the sediments. However, for this block model they were assumed to be barren and grades were not estimated.

Blocks were coded as: 11 = post-mineral volcanics, 20 = pre-mineral volcanics, 31 = sediments. Unassigned blocks do exist outside of the interpreted area. The blocks that did not receive a rock type assignment were assigned a default average density in case they were mined as waste within the Whittle optimized pit shell or mine plan.

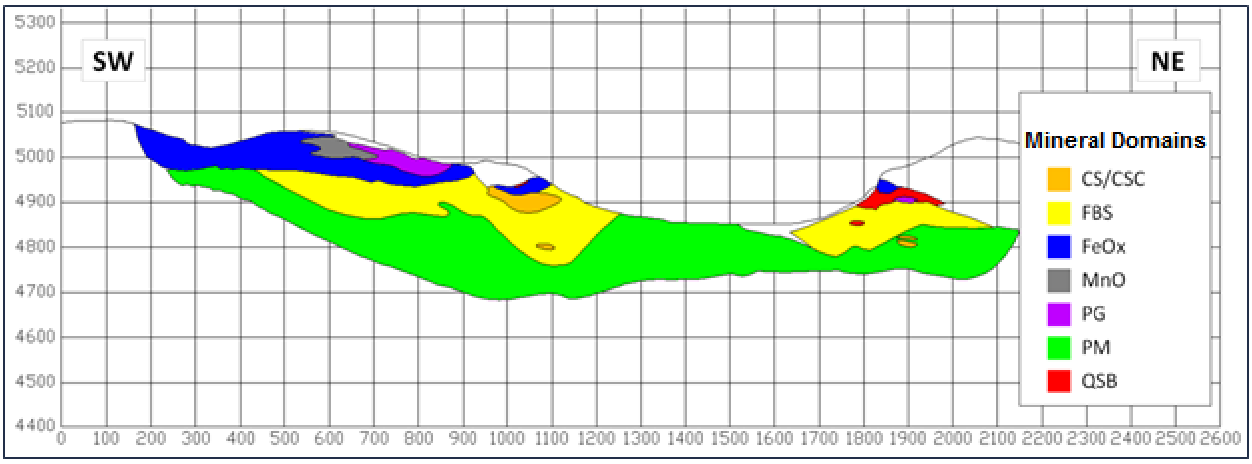
As part of the resource estimation process, BCM updated cross-sections of the drill hole data in each of the deposits and redrew the mineralization boundaries for each of the mineralization domains within the Corani pre-mineral rock units, with the intent of using the solids as domains for grade and recovery estimation.

GRE used the drill hole data, cross-sections with geologic mineralization polygons, geologic logs, and a previously developed and coded block model containing codes for sediments, pre- and post-mineral tuff, and overburden/bofedal provided by BCM. This data and the previous block model codes were verified, and solids were created from mineralization polygons. Geologic log data were also entered in electronic form. The solids were extrapolated a sufficient distance beyond the outside drill holes so as not to limit the ability to assign rock type or grade when grade estimation was completed.

The mineralization solids were checked by generating cross-sections through each of the three areas. The solids were initially used to assign mineralization type to the metallurgical test data. The geometallurgical data analysis is described in more detail in Sections 7 and 13 of this report. Results of that work demonstrated that the mineralization solids were not good predictors of recovery and that a continuous statistical model could be used with better results. Geostatistics of the assay data from the mineralization domains were evaluated, and it was determined that they could be lumped together to form larger, more-contiguous solids, which were used during the estimation of silver, lead, and zinc grades. These groupings generally represent weathered and oxidized rocks, transition or supergene-enriched mineralization, and unaltered sulphide mineralization. The geometallurgical analysis resulted in recovery equations for silver, lead, and zinc, which were estimated for each block of the block model.

Figure 14-2: is an example of the geologic assignment of the mineral domains to the model blocks on cross-section.

Figure 14-2: Geologic Assignment of Mineral Domains to Model Blocks



14.1.2 Density Assignment

Density was based on the combined grade of silver, lead, and zinc. The silver was converted to a percent assay to get common units with lead and zinc. The combined grade and density showed a break at 0.9381%. Blocks with a combined grade under 0.9381% were assigned a density of 2.31 t/m^3, and blocks with a combined grade greater than or equal to 0.9381% were assigned a density of 2.43 t/m^3. Post-mineral tuff with no metal grades to analyse was assigned an average density of 2.3 t/m^3 and non-tuff material was assigned an average density of 2.53 t/m3.

Table 14-2: Assigned Block Densities

Rock Type	Grade	Density
Pre-Mineral Tuff	< 0.9381	2.31
Pre-Mineral Tuff	>= 0.9381	2.43
Post-Mineral Tuff	Not Applicable	2.3
Other Materials	Not Applicable	2.53

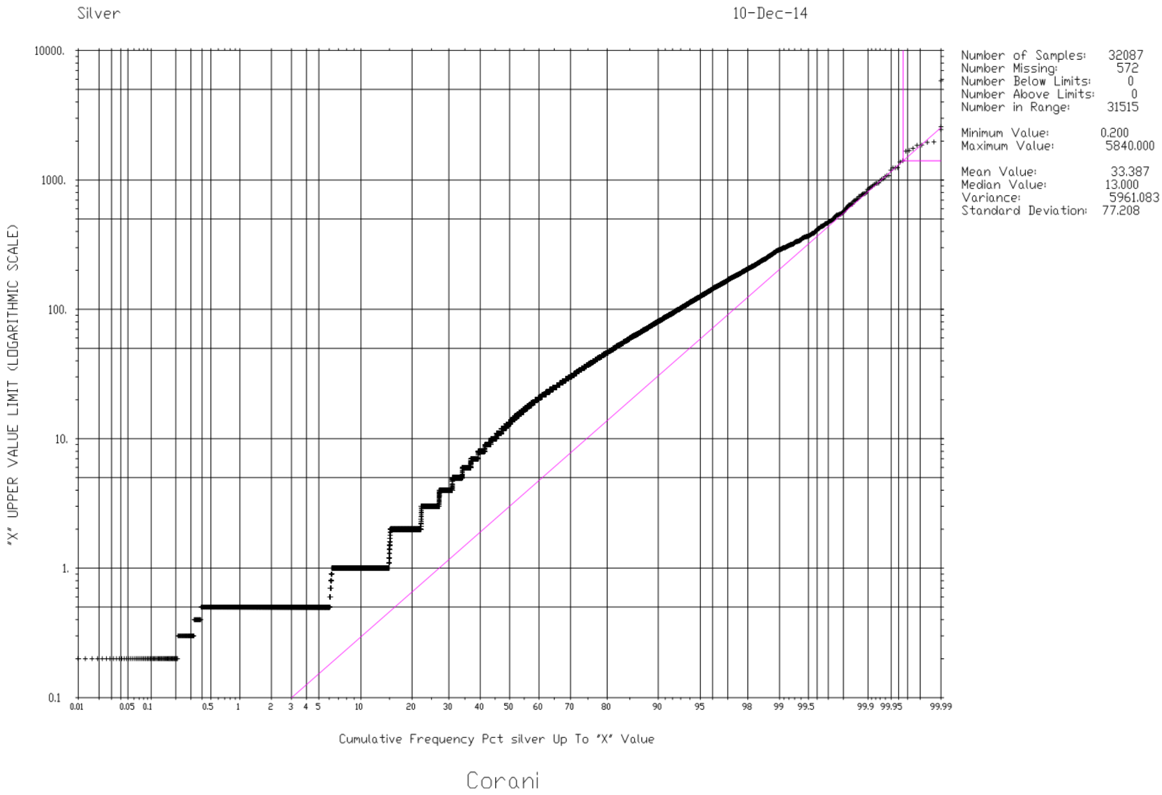
14.1.3 Block Grade Estimation

Block grades were estimated for silver, lead, zinc, copper, goethite, manganese oxide (MnOx), pyrite, and galena. The estimation zones were based on groups of the mineral domains. Coarse sulphides (CS/CSC), fine black sulphides (FBS), pyrite-marcasite (PM), and tetrahedrite (TET) were the first group. Iron oxides (FeOx), manganese oxide (MnOx), and plumbogummite (PG) were the second group. Quartz-sulphide-barite (QSB) was in its own, third group. These members of the groups were selected based on their similar characteristics and location relative to each other. Although QSB had similarities to the first group, it was geographically distinct from the sulphide-rich zones.

Prior to compositing, individual assay values were cut to limit the influence of high-grade outliners on the block grade distribution. The cumulative frequency plot (CFP) of silver has a break at 1,410 ppm, and ten silver samples were capped at 1,410 ppm (Figure 14-3). Although there was a definite break on the copper CFP, it was only used as a recovery indicator, and the break was well above the indicator threshold. Thus, there was no need to cap copper assays. There was no cumulative frequency analysis on the goethite, MnOx, pyrite, and galena labels since there was no assay on those labels.

The cumulative frequency plot of lead manifests itself as a continuous population with no discernible break. For this reason, lead was not capped before compositing. The CFP of zinc shows a continuous population similar to lead; therefore, it was not capped before compositing.

Figure 14-3: Cumulative Frequency Plot of Silver Grade (g/tonne) with Break and Capping



The Corani drill holes were composited to nominal 8-m down-hole or length composites respecting rock type. The composite lengths were slightly modified within each rock type to have composites of equal length that respected lithologic boundaries.

The pre-mineral tuff is the unit of importance for potentially economic mineralization, and can be used as the example of the compositing process. Within each drill hole, the length of the intercept of pre-mineral tuff was determined, and that length was divided into equal-length composites of approximately 8 m. All composites within a rock type in the hole have the same length. That length could be something slightly more or less than the 8-m target value to define the rock type into an integer number of composites. This process eliminates the existence of a short or partial composite at the rock boundary.

The 8-m value was selected based on a grade-versus-count evaluation of alternative composite lengths, to determine whether there was an improvement in ore selectivity with smaller composites versus the cost of production, compared with more cost-efficient use of longer composites and corresponding higher bench heights. The final selection of 8 m was also guided by the expected production rates and mine loading equipment that might be employed for production.

Composite samples for silver, lead, zinc, copper, goethite, MnOx, pyrite, and galena were loaded into SAGE 2001 for variogram analysis and modeling. The composites were divided into groups for each resource area (Main, Minas, and Este) and subdivided into mineral groups. Group 1 consists of the following mineral domains: 1 – CS – Coarse Sulfide; 2 – CSC – Coarse Sulfide and Celadonite; 3 – FBS – Fine Black Sulfide; 8 – PM – Pyrite Marcasite; and 10 – TET – Tetrahedrite. Group 2 consists of these mineral domains: 4 – FeOX – Iron Oxide; 6 – MnO – Manganese Oxide; and 7 – PG – Plumbogummite. Group 3 consists of 9 – QSB – Crystalline quartz sulfide barite. These groups were created based on the relative locations of the mineral domains to each other. This created 9 composite groups for the 8 grade labels, resulting in 72 total variograms to model.

It was determined that a correlogram created the best fit for this set of data. Downhole correlograms were used to determine the nugget for each composite sample group. Experimental correlograms had search parameters set for 30-degree increments for azimuth and 15-degree increments for dip. Omnidirectional correlograms were created for grade zones that did not have a distinct directional trend. These particular grade zones were modeled with isotropic parameters. The rest were modeled directionally to create anisotropic parameters for block modeling.

Typically, the general trend of all the estimation zones followed the trend of the contact between pre-mineral tuff and post-mineral tuff, which is dip direction of 300 to 330 degrees azimuth and dipping 15 to 30 degrees down. This is in agreement with previous authors and BCM geologists. Silver, with a mobility different from lead and zinc, was estimated across mineral domain boundaries. The evidence of high silver grades in areas of expected low sulphidization and a smooth gradient of grades across mineral boundaries points to this as an appropriate modeling method. The method chosen for all grades was Inverse Distance to a Power (IDP), specifically ID3 for silver and ID2.5 for all others. This method was picked to model the high local variability of the drill hole data. By adjusting the power, a satisfactory level of smoothness was applied to the grade model. The IDP model was compared to a kriged model, and IDP was chosen due to the kriged model's failure to highlight the highest and lowest grades in the population.

Figure 14-4 through Figure 14-24 present the correlograms for the three economically viable metals in each deposit zone and mineral domain (where applicable). Table 14-3 through Table 14-12 show the parameters for input to the modeling software for all eight grade labels in the block model. These parameters were taken from the variography of the composite data. Notably, the search range was increased from the previous study. The range from the variography analysis showed that the model has a range of 180 m in two zones and 170 m in another zone for the silver models. Since silver is the primary metal in this project, a search range of 180 m was used for all grade labels to preserve uniformity of the number of blocks modeled.

Figure 14-4: Silver Correlogram – Este

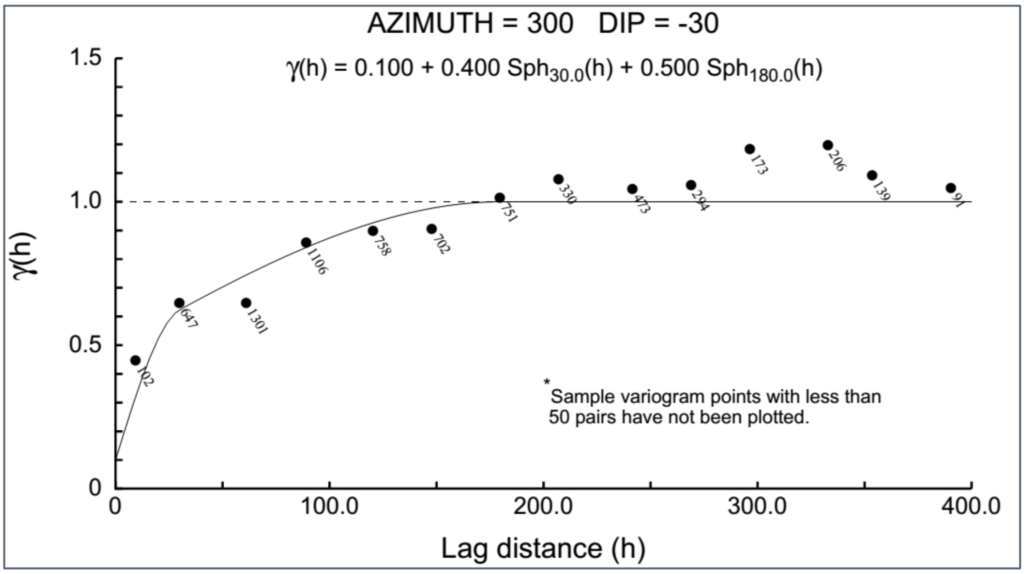


Figure 14-5: Silver Correlogram – Main

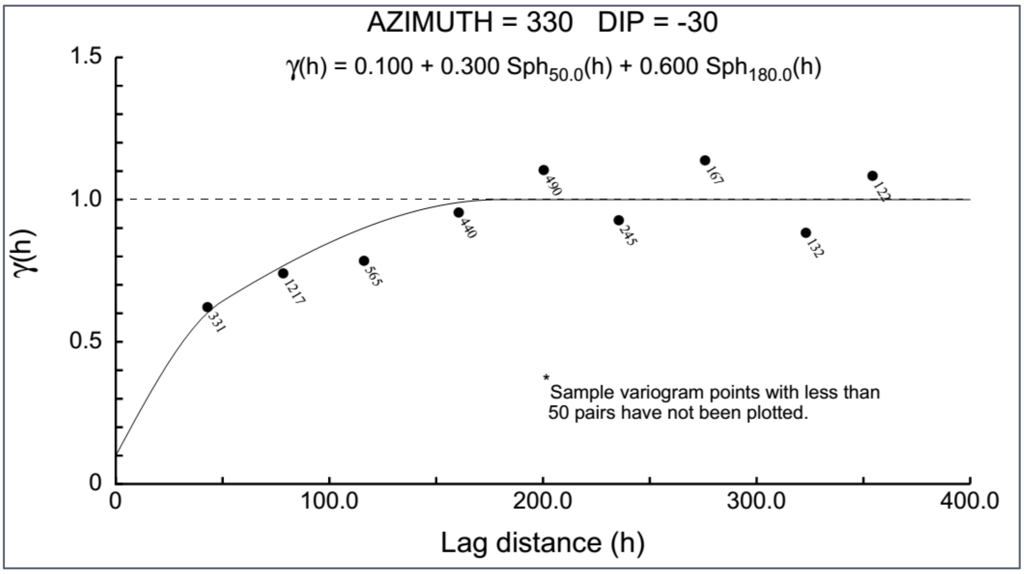


Figure 14-6: Silver Correlogram – Minas

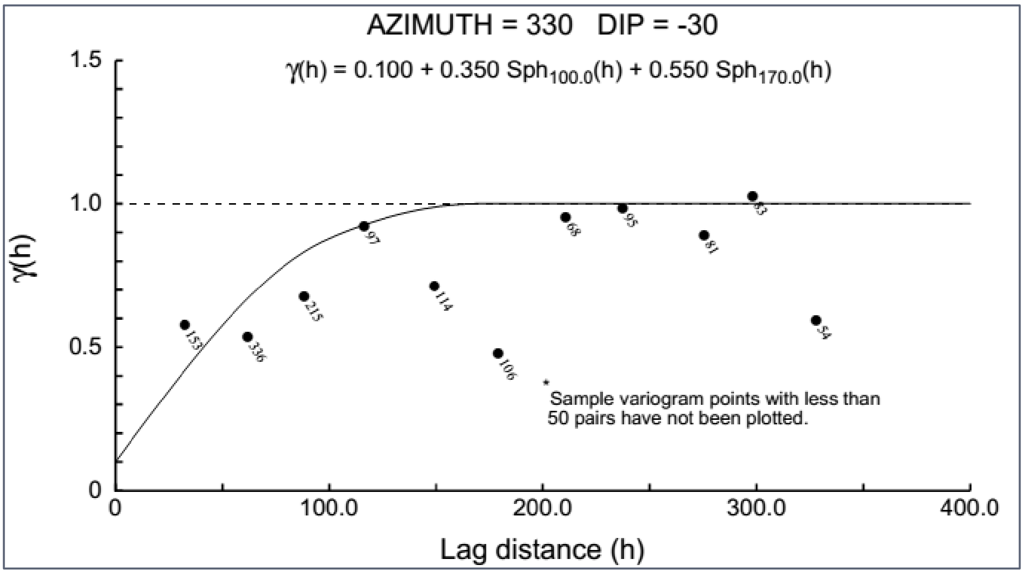


Figure 14-7: Lead Correlogram – Main, Mineral Group 1

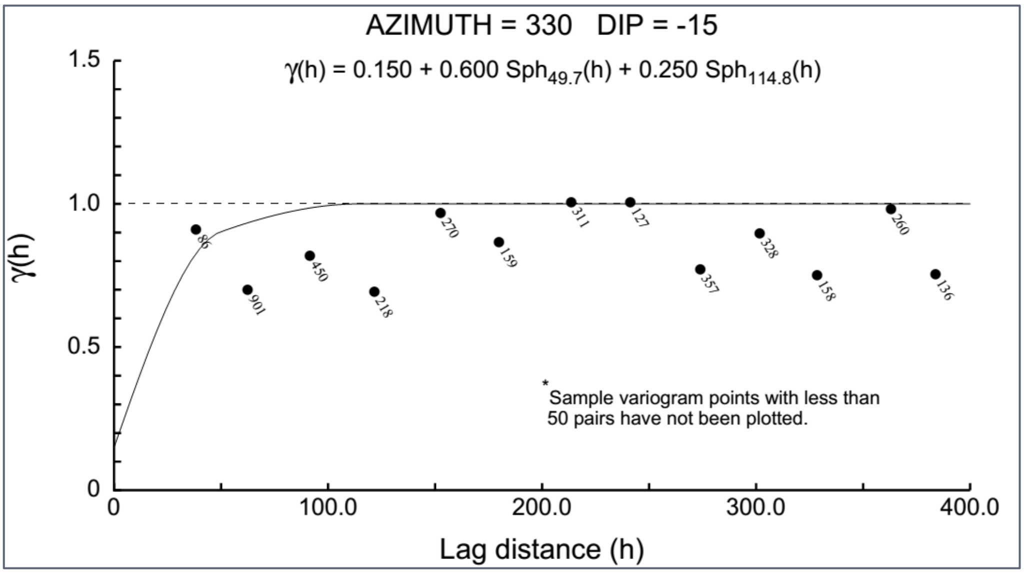


Figure 14-8: Lead Correlogram – Main, Mineral Group 2

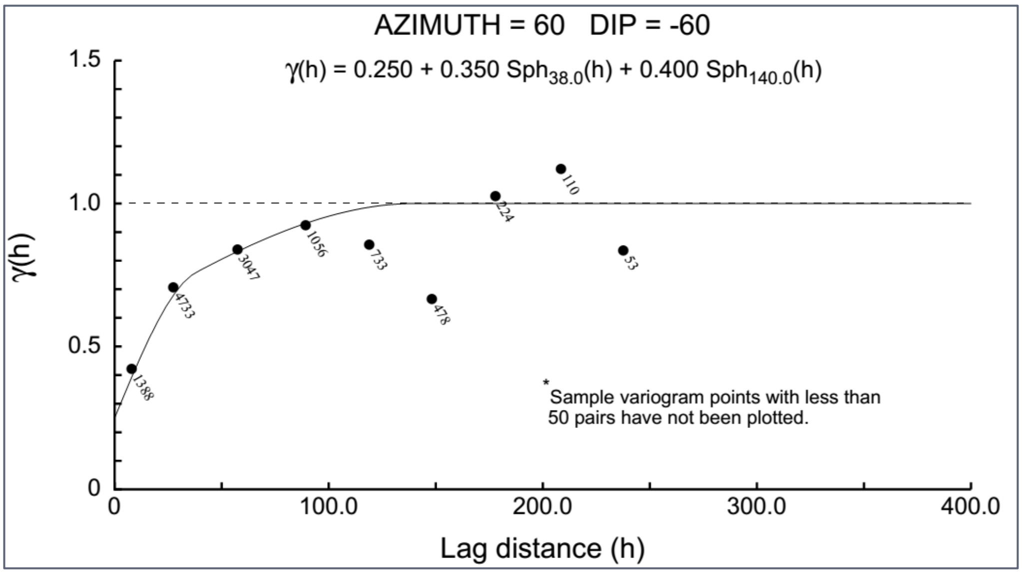


Figure 14-9: Lead Correlogram – Main, Mineral Group 3

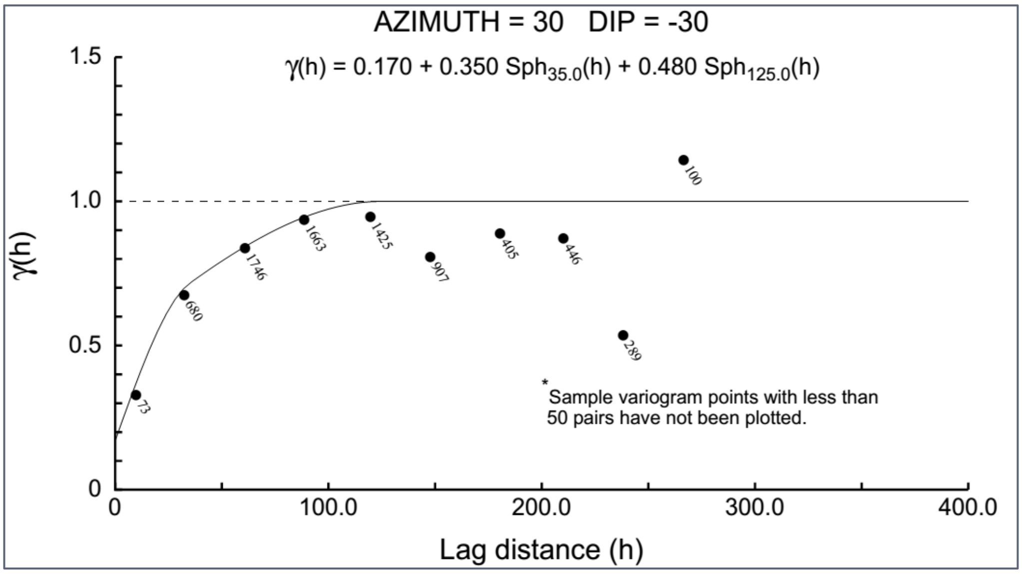


Figure 14-10: Lead Correlogram – Minas, Mineral Group 1

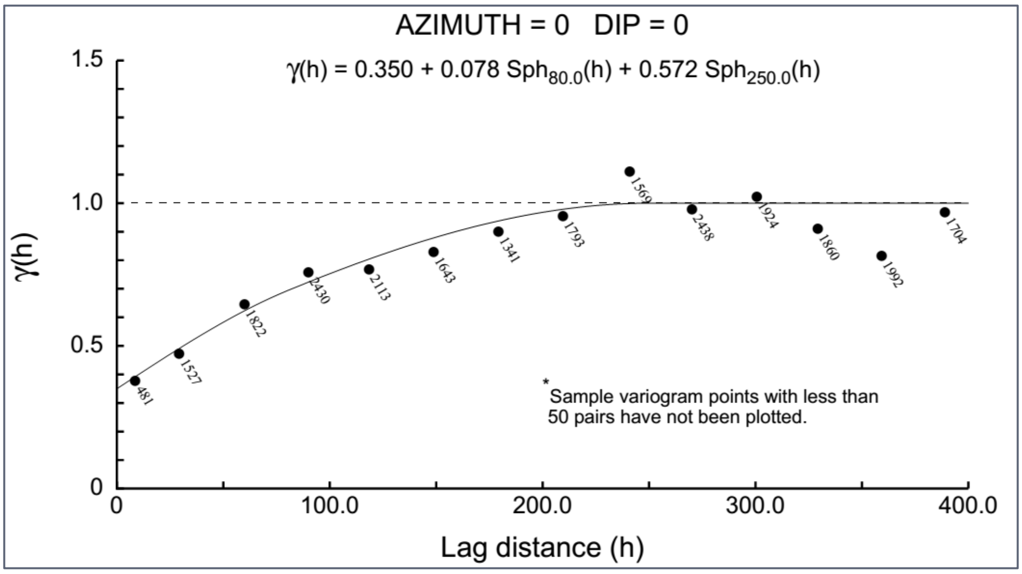


Figure 14-11: Lead Correlogram – Minas, Mineral Group 2

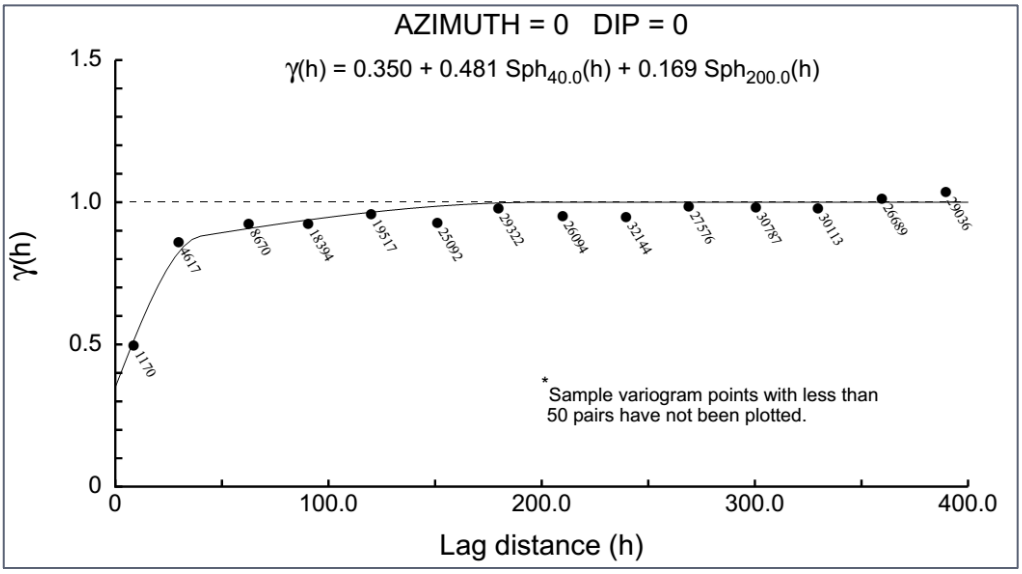


Figure 14-12: Lead Correlogram – Minas, Mineral Group 3

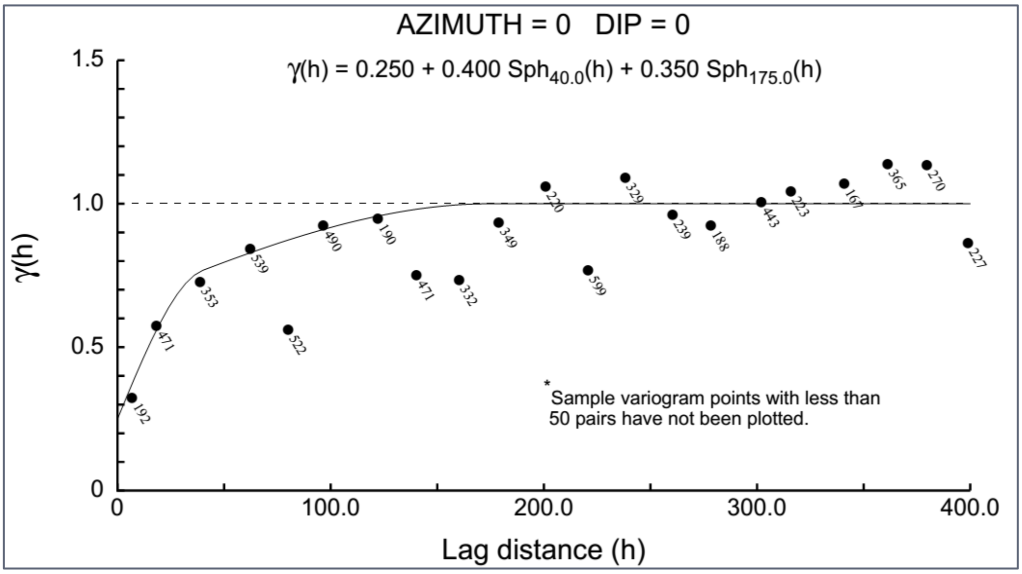


Figure 14-13: Lead Correlogram – Este, Mineral Group 1

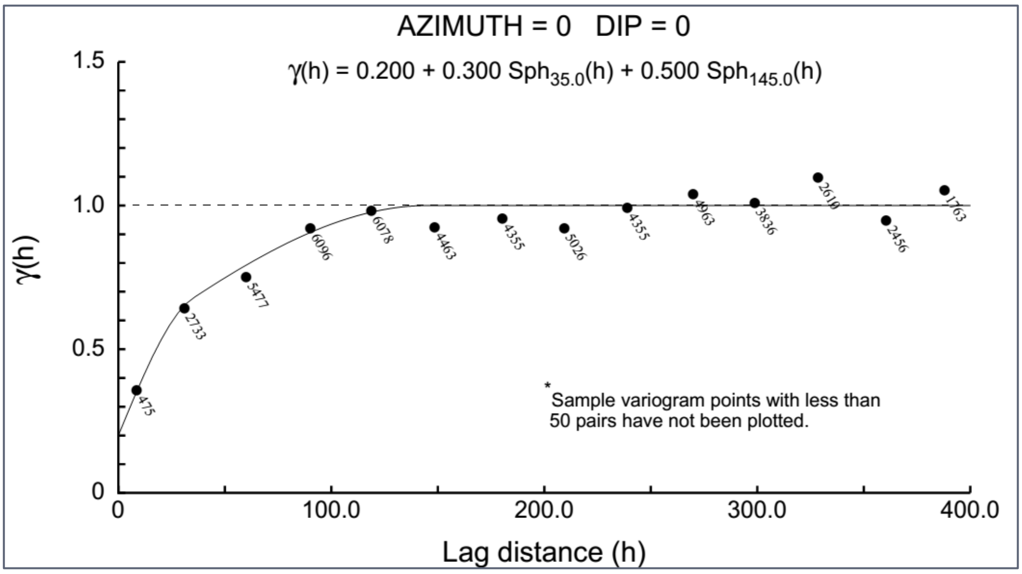


Figure 14-14: Lead Correlogram – Este, Mineral Group 2

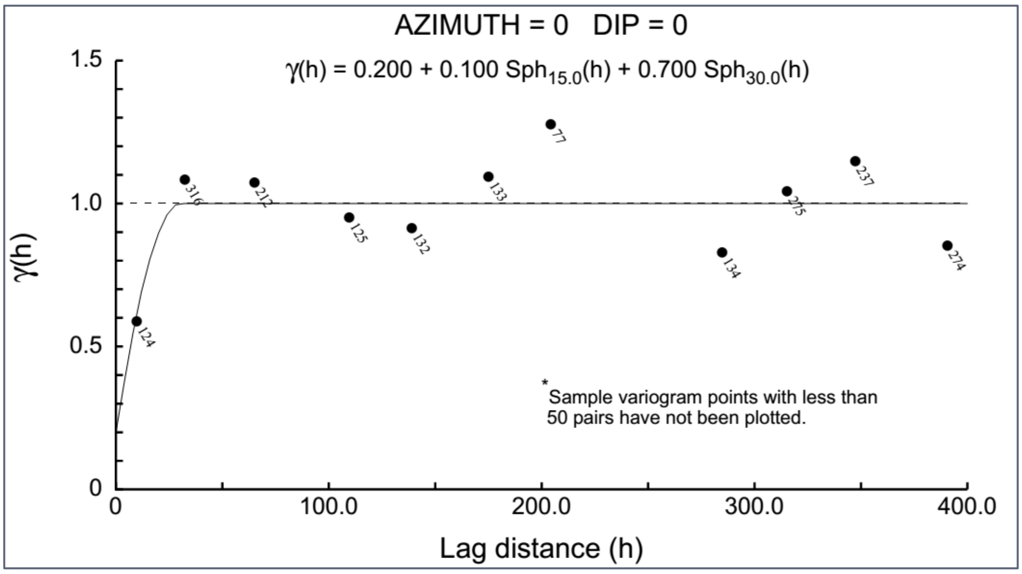


Figure 14-15: Lead Correlogram – Este, Mineral Group 3

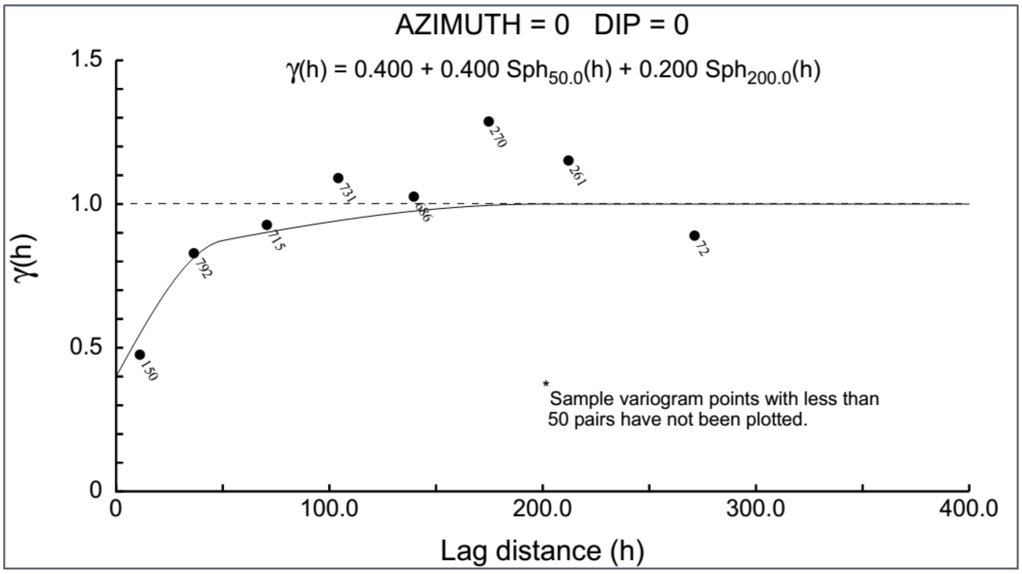


Figure 14-16: Zinc Correlogram – Main, Mineral Group 1

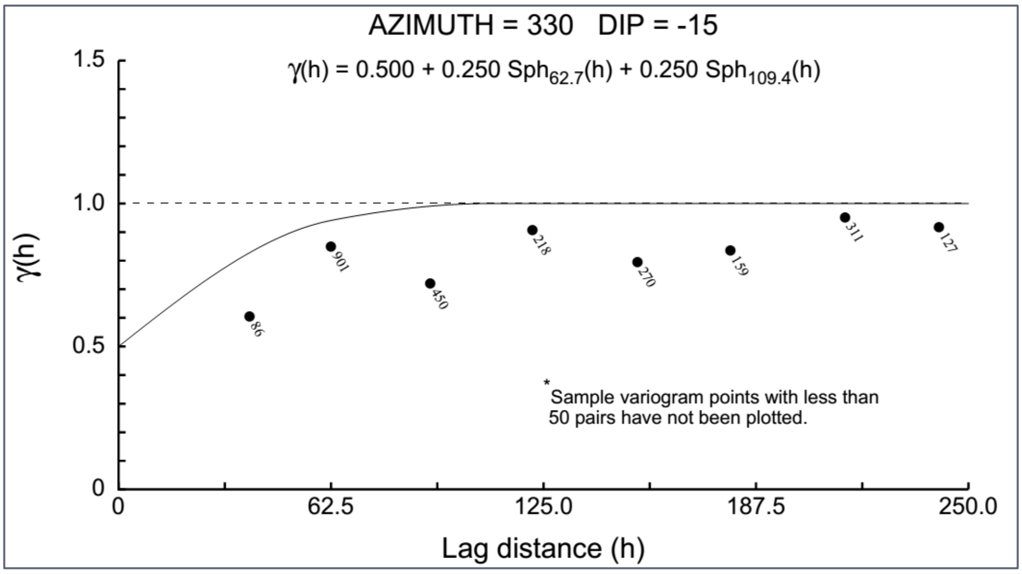


Figure 14-17: Zinc Correlogram – Main, Mineral Group 2

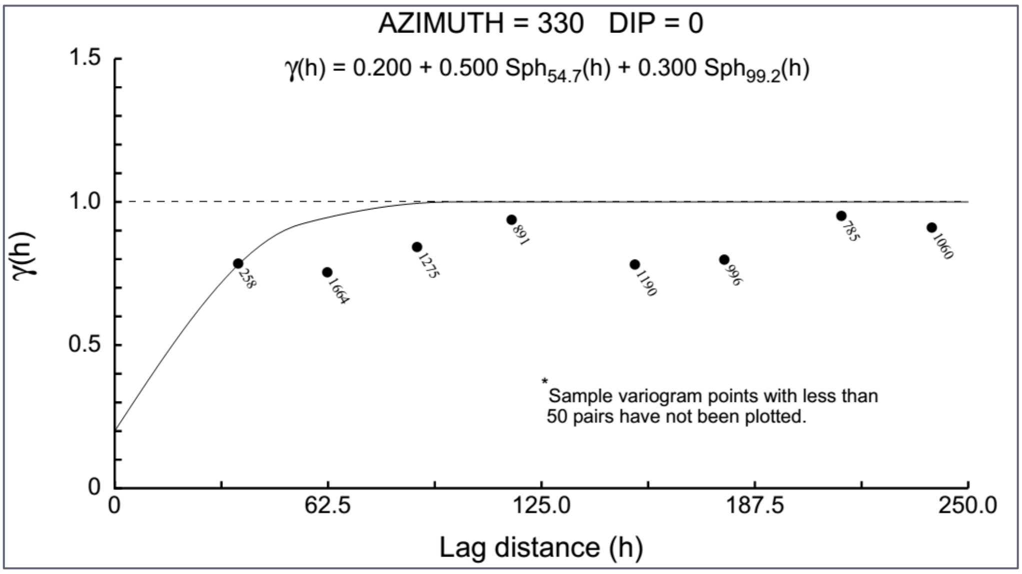


Figure 14-18: Zinc Correlogram – Main, Mineral Group 3

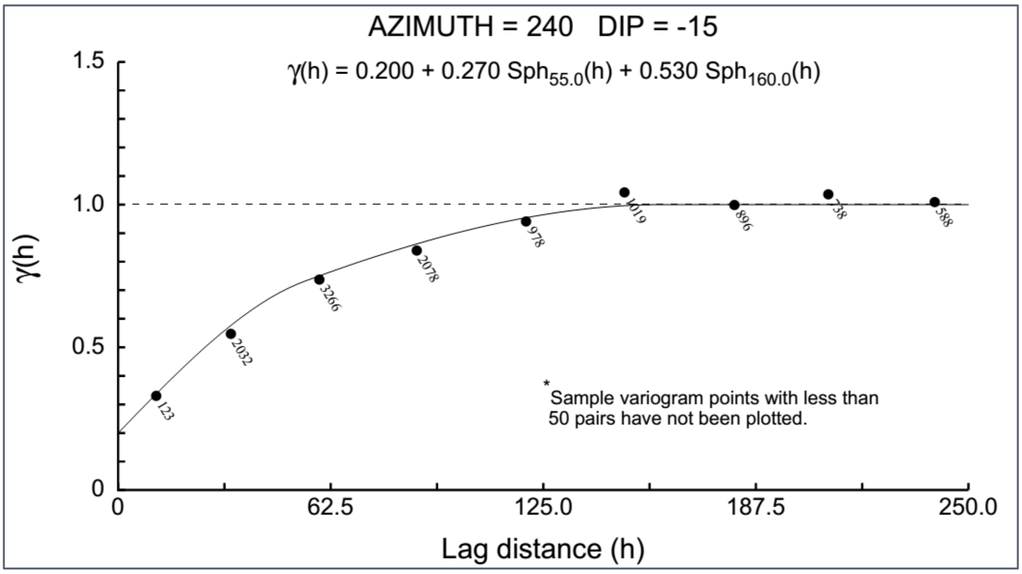


Figure 14-19: Zinc Correlogram – Minas, Mineral Group 1

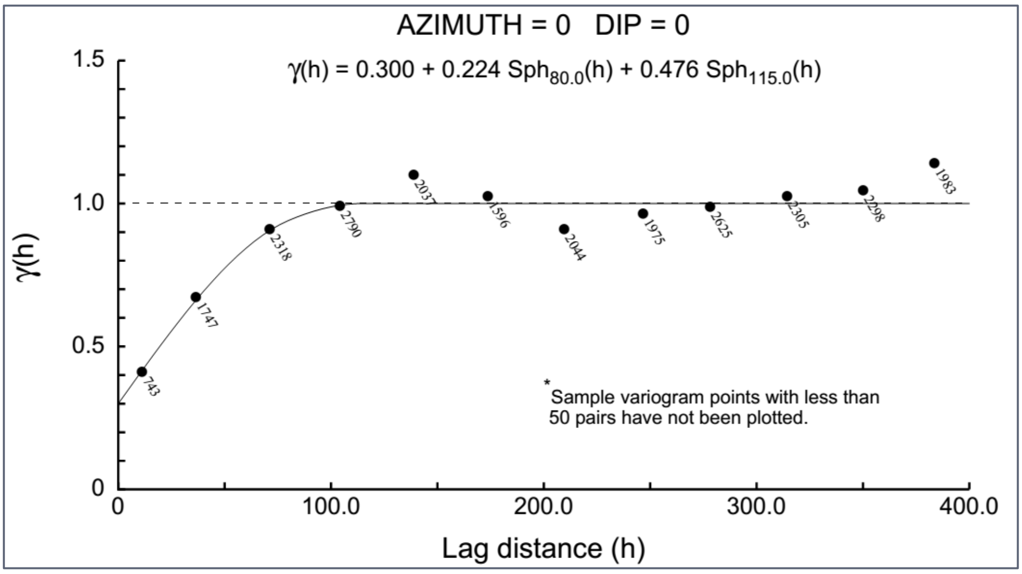


Figure 14-20: Zinc Correlogram – Minas, Mineral Group 2

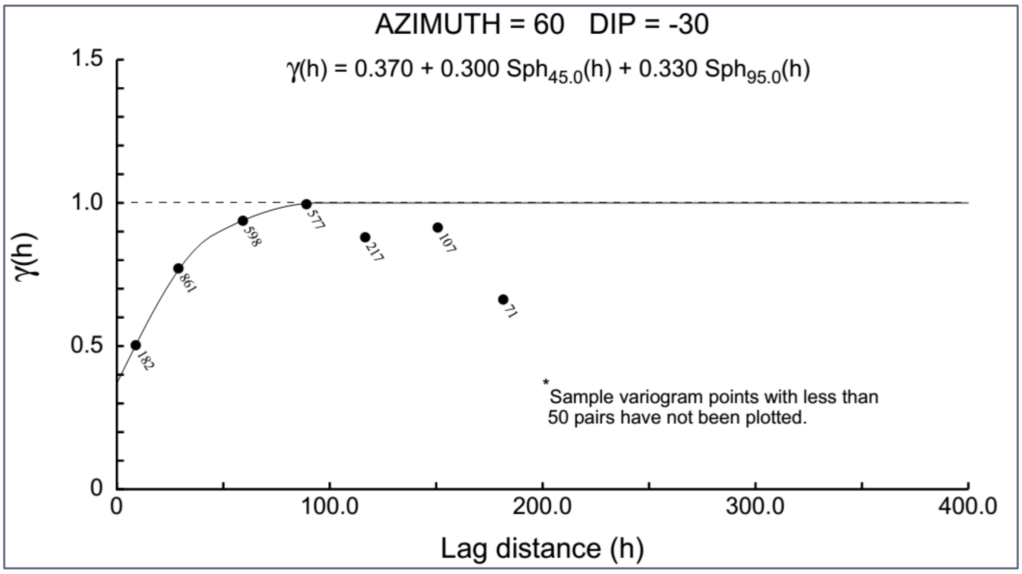


Figure 14-21: Zinc Correlogram – Minas, Mineral Group 3

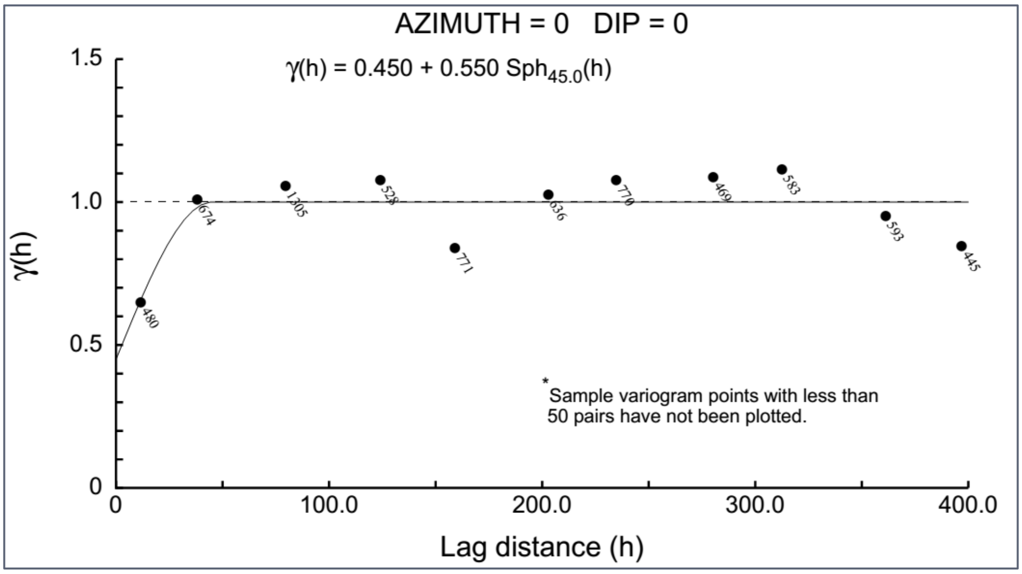


Figure 14-22: Zinc Correlogram – Este, Mineral Group 1

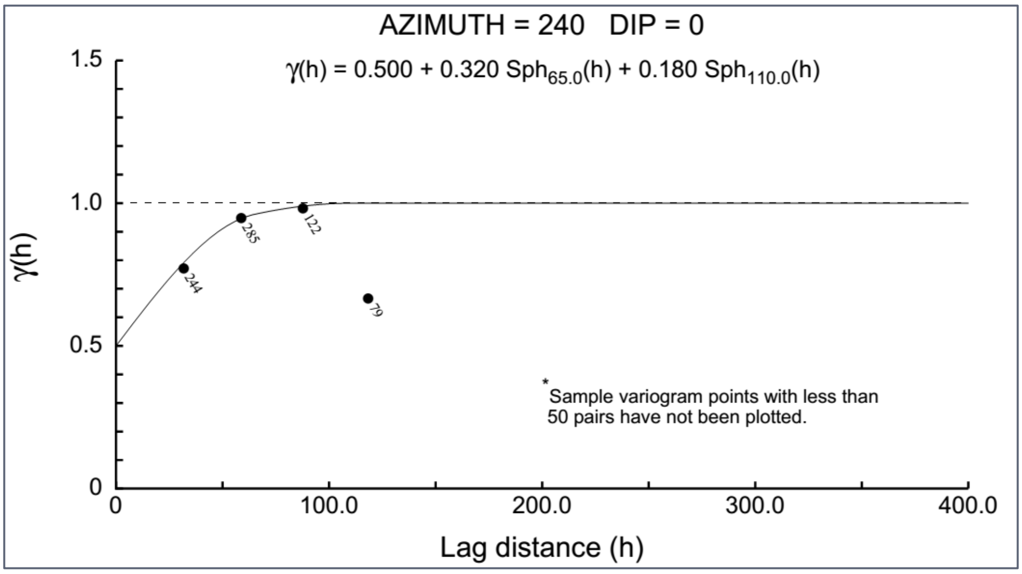


Figure 14-23: Zinc Correlogram – Este, Mineral Group 2

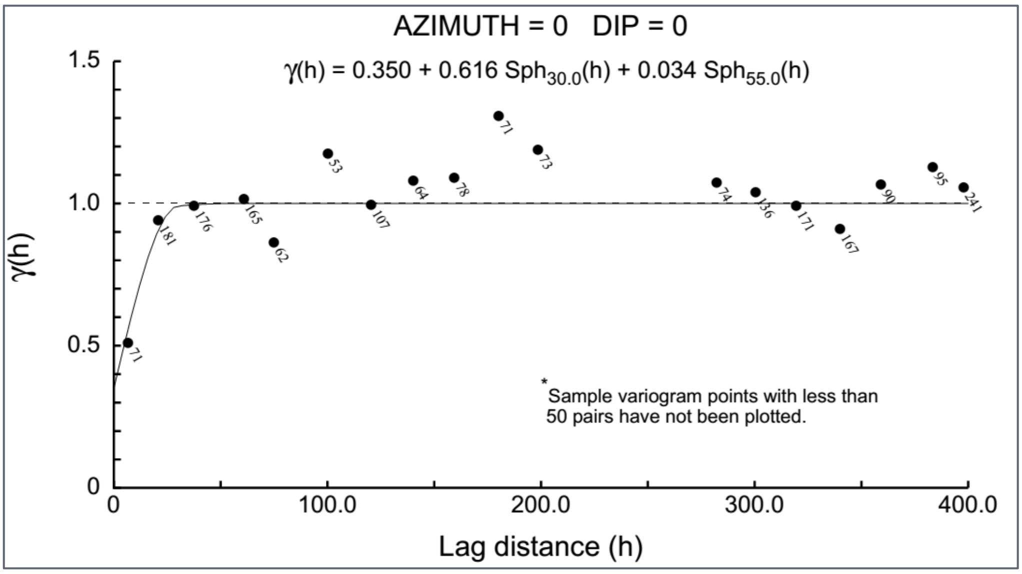


Figure 14-24: Zinc Correlogram – Este, Mineral Group 3

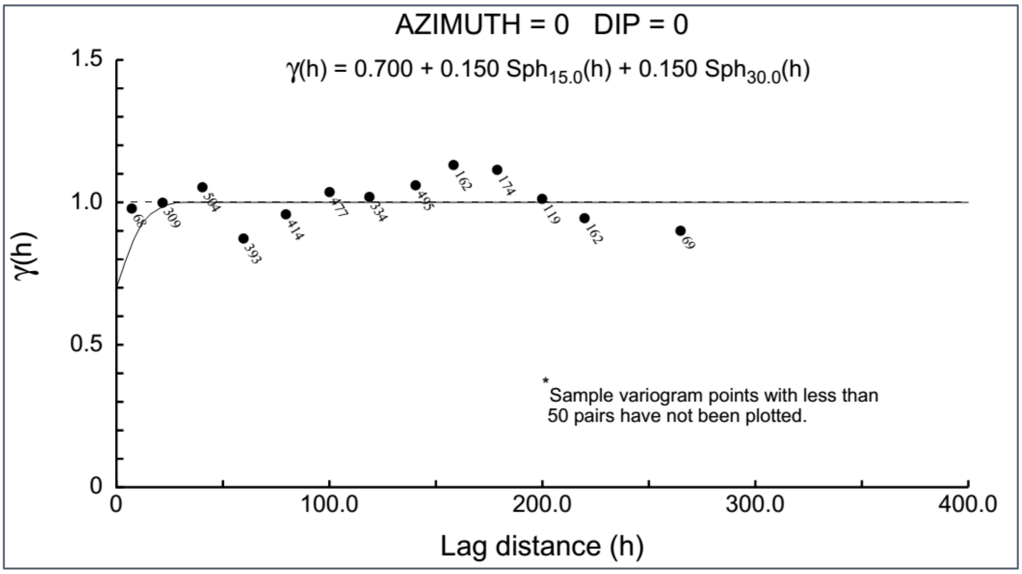


Table 14-3: Modeling Parameters for Silver, Lead, and Zinc

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Silver	Main	ALL	0.1	0.35	0.55	100	170	330	-30	15	170	140	100	180
Silver	Minas	ALL	0.1	0.3	0.6	50	180	330	-30	15	180	150	90	180
Silver	Este	ALL	0.1	0.4	0.5	30	180	330	-30	15	180	150	95	180
Lead	Main	1 2 3 8 10	0.15	0.6	0.25	50	125	338	-14	30	125	120	55	180
Lead	Main	4 6 7	0.25	0.35	0.4	38	145	330	0	-60	145	140	100	180
Lead	Main	9	0.17	0.35	0.48	35	130	334	-14	15	130	125	70	180
Lead	Minas	1 2 3 8 10	0.35	0.08	0.57	80	250	0	0	0	250	250	250	180
Lead	Minas	4 6 7	0.35	0.48	0.17	40	200	0	0	0	200	200	200	180
Lead	Minas	9	0.25	0.35	0.4	40	175	0	0	0	175	175	175	180
Lead	Este	1 2 3 8 10	0.2	0.3	0.5	35	145	0	0	0	145	145	145	180
Lead	Este	4 6 7	0.2	0.1	0.7	15	30	0	0	0	30	30	30	180
Lead	Este	9	0.4	0.4	0.2	50	200	0	0	0	200	200	200	180
Zinc	Main	1 2 3 8 10	0.5	0.25	0.25	50	140	338	-14	30	140	135	100	180

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Zinc	Main	4 6 7	0.2	0.5	0.3	45	125	330	0	-60	125	120	90	180
Zinc	Main	9	0.2	0.27	0.53	55	165	334	-14	15	165	160	60	180
Zinc	Minas	1 2 3 8 10	0.3	0.22	0.48	80	115	0	0	0	115	115	115	180
Zinc	Minas	4 6 7	0.37	0.3	0.33	45	95	60	-30	0	95	80	70	180
Zinc	Minas	9	0.45	0.55	0	45	45	0	0	0	45	45	45	180
Zinc	Este	1 2 3 8 10	0.5	0.32	0.18	65	110	240	0	0	110	60	60	180
Zinc	Este	4 6 7	0.35	0.62	0.03	30	55	0	0	0	55	55	55	180
Zinc	Este	9	0.7	0.15	0.15	15	30	0	0	0	30	30	30	180

Table 14-4: Modeling Parameters for Copper, Geothite, and MnOx

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Copper	Main	1 2 3 8 10	0.6	0.05	0.35	100	180	330	-15	15	180	150	45	180
Copper	Main	4 6 7	0.4	0.45	0.15	75	150	330	-30	30	150	125	75	180
Copper	Main	9	0.5	0.1	0.4	75	220	330	-15	15	220	215	60	180
Copper	Minas	1 2 3 8 10	0.4	0.32	0.28	30	275	0	0	0	275	275	275	180
Copper	Minas	4 6 7	0.4	0.1	0.5	75	200	330	-15	15	200	90	50	180
Copper	Minas	9	0.3	0.24	0.46	40	250	0	0	0	250	250	250	180
Copper	Este	1 2 3 8 10	0.5	0.2	0.3	70	220	330	-15	30	220	125	40	180
Copper	Este	4 6 7	0.3	0.42	0.28	30	140	0	0	0	140	140	140	180
Copper	Este	9	0.3	0.38	0.32	35	135	0	0	0	135	135	135	180
Goethite	Main	1 2 3 8 10	0.45	0.25	0.3	55	120	0	0	0	120	120	120	180
Goethite	Main	4 6 7	0.35	0.55	0.1	50	125	240	-45	-85	125	120	80	180
Goethite	Main	9	0.45	0.25	0.3	50	110	270	-30	-10	110	100	60	180
Goethite	Minas	1 2 3 8 10	0.25	0.6	0.15	40	100	270	-60	30	100	50	90	180
Goethite	Minas	4 6 7	0.1	0.6	0.3	60	130	240	-30	0	130	55	120	180
Goethite	Minas	9	0.65	0.16	0.19	80	160	0	0	0	160	160	160	180
Goethite	Este	1 2 3 8 10	0.25	0.37	0.38	30	125	90	-15	0	125	90	75	180
Goethite	Este	4 6 7	0.4	0.37	0.23	10	65	0	0	0	65	65	65	180
Goethite	Este	9	0.15	0.85	0	40	40	0	0	0	40	40	40	180
MnOx	Main	1 2 3 8 10	0.03	0.17	0.8	40	250	30	-60	-70	250	165	120	180
MnOx	Main	4 6 7	0.2	0.7	0.1	40	125	60	0	0	125	85	110	180
MnOx	Main	9	0.2	0.3	0.5	110	210	240	-15	0	210	60	200	180
MnOx	Minas	1 2 3 8 10	0.1	0.55	0.35	60	175	0	0	0	175	175	175	180
MnOx	Minas	4 6 7	0.1	0.8	0.1	40	125	240	-15	0	125	100	100	180
MnOx	Minas	9	0.35	0.05	0.6	20	225	0	0	0	225	225	225	180
MnOx	Este	1 2 3 8 10	0.1	0.2	0.7	75	135	300	-60	30	135	75	125	180
MnOx	Este	4 6 7	0.35	0.65	0	66	66	0	0	0	66	66	66	180
MnOx	Este	9	0.6	0.4	0	50	50	0	0	0	50	50	50	180

Table 14-5: Modeling Parameters for Pyrite and Galena

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Pyrite	Main	1 2 3 8 10	0.3	0.15	0.55	30	150	240	-45	0	150	55	120	180
Pyrite	Main	4 6 7	0.3	0.3	0.4	20	95	240	-45	-25	95	95	75	180
Pyrite	Main	9	0.2	0.45	0.35	45	90	240	-45	-65	90	90	75	180
Pyrite	Minas	1 2 3 8 10	0.1	0.32	0.58	30	350	0	0	0	350	350	350	180
Pyrite	Minas	4 6 7	0.65	0.24	0.11	31	150	0	0	0	150	150	150	180
Pyrite	Minas	9	0.72	0.12	0.16	98	100	0	0	0	100	100	100	180

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Pyrite	Este	1 2 3 8 10	0.15	0.64	0.21	42	125	0	0	0	125	125	125	180
Pyrite	Este	4 6 7	0.4	0.6	0	40	40	0	0	0	40	40	40	180
Pyrite	Este	9	0.05	0.75	0.2	35	40	0	0	0	40	40	40	180
Galena	Main	1 2 3 8 10	0.3	0.5	0.2	33	125	240	-30	-15	125	100	110	180
Galena	Main	4 6 7	0.3	0.42	0.28	45	170	60	-45	-60	170	75	50	180
Galena	Main	9	0.2	0.45	0.35	55	135	150	-15	-75	135	65	105	180
Galena	Minas	1 2 3 8 10	0.75	0.14	0.11	35	80	0	0	0	80	80	80	180
Galena	Minas	4 6 7	0.25	0.58	0.17	25	100	0	0	0	100	100	100	180
Galena	Minas	9	0.7	0.25	0.05	8	80	0	0	0	80	80	80	180
Galena	Este	1 2 3 8 10	0.4	0.6	0	60	60	0	0	0	60	60	60	180
Galena	Este	4 6 7	0.8	0.14	0.06	5	50	0	0	0	50	50	50	180
Galena	Este	9	0.2	0.25	0.55	20	50	0	0	0	50	50	50	180

Table 14-6: Borehole Statistics for Silver, Lead, and Zinc

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	1	1	4	70.0	187.0	125.5	2,140.9	46.3	0.4
Ag	1	2	376	0.3	1,410.0	86.9	23,167.0	152.2	1.8
Ag	1	3	11,791	0.2	2,580.0	42.1	6,599.1	81.2	1.9
Ag	1	8	11,452	0.2	5,840.0	13.8	4,276.0	65.4	4.7
Ag	1	10	5	80.0	364.0	149.4	13,122.0	114.6	0.8
Ag	1	ALL	23,628	0.2	5,840.0	29.1	5,991.7	77.4	2.7
Ag	2	4	3,341	0.3	1,240.0	36.3	4,945.9	70.3	1.9
Ag	2	6	814	0.5	243.0	44.5	1,777.2	42.2	0.9
Ag	2	7	1,154	1.0	420.0	40.4	1,286.8	35.9	0.9
Ag	2	ALL	5,309	0.3	1,240.0	38.5	3,680.2	60.7	1.6
Ag	3	9	2,019	0.5	1,750.0	63.2	5,810.8	76.2	1.2
Ag	0	ALL	34,649	0.2	5,840.0	33.2	5,660.6	75.2	2.3
Pb	1	1	4	1.72	3.54	2.87	0.54	0.73	0.26
Pb	1	2	376	0.01	16.65	2.20	4.93	2.22	1.01
Pb	1	3	11,722	0.00	15.55	0.71	1.00	1.00	1.40
Pb	1	8	11,278	0.00	15.30	0.31	0.47	0.68	2.22
Pb	1	10	5	1.25	2.17	1.75	0.11	0.33	0.19
Pb	1	ALL	23,385	0.00	16.65	0.54	0.89	0.94	1.74
Pb	2	4	3,324	0.00	5.93	0.42	0.38	0.62	1.46
Pb	2	6	814	0.01	10.85	0.77	1.03	1.02	1.32
Pb	2	7	1,154	0.01	5.92	0.67	0.55	0.74	1.11
Pb	2	ALL	5,292	0.00	10.85	0.53	0.54	0.73	1.38
Pb	3	9	2,019	0.01	8.92	0.94	0.93	0.97	1.02
Pb	0	ALL	34,190	0.00	16.65	0.56	0.82	0.90	1.62
Zn	1	1	4	0.78	2.17	1.80	0.40	0.63	0.35
Zn	1	2	376	0.04	9.00	1.03	1.91	1.38	1.34
Zn	1	3	11,677	0.00	16.15	0.42	1.01	1.00	2.38
Zn	1	8	11,430	0.00	23.91	0.48	0.73	0.86	1.80
Zn	1	10	5	0.11	0.20	0.16	0.00	0.04	0.24
Zn	1	ALL	23,492	0.00	23.91	0.46	0.89	0.95	2.07
Zn	2	4	3,211	0.01	3.45	0.10	0.02	0.15	1.49
Zn	2	6	808	0.01	5.03	0.18	0.06	0.24	1.32
Zn	2	7	1,137	0.01	1.14	0.13	0.01	0.11	0.82

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Zn	2	ALL	5,156	0.01	5.03	0.12	0.03	0.17	1.35
Zn	3	9	1,904	0.01	3.24	0.15	0.05	0.23	1.49
Zn	0	ALL	33,313	0.00	23.91	0.37	0.67	0.82	2.24

Table 14-7: Composite Statistics for Silver, Lead, and Zinc

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	1	1	1	125.5	125.5	125.5	0.0	0.0	0.0
Ag	1	2	101	2.6	461.8	83.4	7,517.5	86.7	1.0
Ag	1	3	2,993	0.2	730.9	41.7	3,061.4	55.3	1.3
Ag	1	8	2,917	0.4	704.0	13.3	791.9	28.1	2.1
Ag	1	10	1	181.4	181.4	181.4	0.0	0.0	0.0
Ag	1	ALL	6,013	0.2	730.9	28.6	2,287.8	47.8	1.7
Ag	2	4	857	0.5	534.1	36.7	3,400.2	58.3	1.6
Ag	2	6	212	1.0	166.4	43.9	1,216.5	34.9	0.8
Ag	2	7	298	2.0	205.0	40.2	761.3	27.6	0.7
Ag	2	ALL	1,367	0.5	534.1	38.6	2,491.4	49.9	1.3
Ag	3	9	525	1.2	472.6	63.0	3,041.4	55.1	0.9
Ag	0	ALL	8,872	0.2	730.9	33.4	2,598.8	51.0	1.5
Pb	1	1	1	2.87	2.87	2.87	0.00	0.00	0.00
Pb	1	2	101	0.07	8.22	2.15	2.34	1.53	0.71
Pb	1	3	2,993	0.00	8.51	0.71	0.62	0.79	1.11
Pb	1	8	2,908	0.00	6.97	0.30	0.26	0.51	1.71
Pb	1	10	1	2.17	2.17	2.17	0.00	0.00	0.00
Pb	1	ALL	6,004	0.00	8.51	0.53	0.56	0.75	1.40
Pb	2	4	857	0.00	3.44	0.42	0.28	0.53	1.26
Pb	2	6	212	0.01	5.75	0.77	0.76	0.87	1.14
Pb	2	7	298	0.01	3.46	0.69	0.39	0.62	0.91
Pb	2	ALL	1,367	0.00	5.75	0.53	0.40	0.63	1.19
Pb	3	9	525	0.02	7.05	0.94	0.60	0.77	0.83
Pb	0	ALL	8,841	0.00	8.51	0.56	0.54	0.73	1.32
Zn	1	1	1	1.80	1.80	1.80	0.00	0.00	0.00
Zn	1	2	101	0.12	7.89	1.05	1.24	1.11	1.06
Zn	1	3	2,965	0.00	7.48	0.42	0.65	0.80	1.91
Zn	1	8	2,912	0.01	9.75	0.47	0.43	0.66	1.39
Zn	1	10	1	0.15	0.15	0.15	0.00	0.00	0.00
Zn	1	ALL	5,980	0.00	9.75	0.46	0.56	0.75	1.64
Zn	2	4	816	0.01	2.24	0.10	0.02	0.13	1.25
Zn	2	6	210	0.02	1.26	0.17	0.02	0.15	0.89
Zn	2	7	293	0.01	0.80	0.13	0.01	0.09	0.69
Zn	2	ALL	1,319	0.01	2.24	0.12	0.02	0.13	1.06
Zn	3	9	493	0.01	1.82	0.15	0.03	0.17	1.13
Zn	0	ALL	8,500	0.00	9.75	0.36	0.43	0.66	1.80

Table 14-8: Composite Statistics for Copper, MnOx, and Goethite

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Cu	1	1	1	0.05	0.05	0.05	0.00	0.00	0.00
Cu	1	2	101	0.00	0.27	0.05	0.00	0.05	1.01
Cu	1	3	2,724	0.00	1.05	0.03	0.00	0.05	1.49
Cu	1	8	1,752	0.00	0.54	0.03	0.00	0.05	1.61
Cu	1	10	1	0.06	0.06	0.06	0.00	0.00	0.00
Cu	1	ALL	4,579	0.00	1.05	0.03	0.00	0.05	1.52
Cu	2	4	694	0.00	0.59	0.02	0.00	0.03	1.68
Cu	2	6	210	0.00	0.15	0.03	0.00	0.02	0.81
Cu	2	7	288	0.00	0.08	0.02	0.00	0.01	0.71
Cu	2	ALL	1,192	0.00	0.59	0.02	0.00	0.03	1.30
Cu	3	9	486	0.00	0.22	0.04	0.00	0.03	0.68
Cu	0	ALL	6,800	0.00	1.05	0.03	0.00	0.04	1.45
MnOx	1	1	0	0.00	0.00	0.00	0.00	0.00	0.00
MnOx	1	2	32	0.01	2.00	0.82	0.50	0.71	0.86
MnOx	1	3	806	0.00	5.48	0.72	0.69	0.83	1.15
MnOx	1	8	344	0.00	6.00	0.63	0.59	0.77	1.21
MnOx	1	10	0	0.00	0.00	0.00	0.00	0.00	0.00
MnOx	1	ALL	1,182	0.00	6.00	0.70	0.66	0.81	1.16
MnOx	2	4	260	0.00	4.97	0.62	0.53	0.73	1.17
MnOx	2	6	124	0.00	5.25	0.81	0.96	0.98	1.22
MnOx	2	7	112	0.01	2.67	0.48	0.45	0.67	1.40
MnOx	2	ALL	496	0.00	5.25	0.64	0.63	0.80	1.25
MnOx	3	9	312	0.00	5.39	1.26	0.80	0.89	0.71
MnOx	0	ALL	2,338	0.00	6.00	0.70	0.69	0.83	1.19
Goeth	1	1	0	0.00	0.00	0.00	0.00	0.00	0.00
Goeth	1	2	17	0.01	1.00	0.29	0.10	0.31	1.07
Goeth	1	3	508	0.00	3.00	0.38	0.26	0.51	1.35
Goeth	1	8	95	0.00	5.15	0.41	0.64	0.80	1.95
Goeth	1	10	0	0.00	0.00	0.00	0.00	0.00	0.00
Goeth	1	ALL	620	0.00	5.15	0.38	0.31	0.56	1.47
Goeth	2	4	563	0.00	5.39	0.66	0.63	0.79	1.19
Goeth	2	6	180	0.01	4.37	1.14	0.75	0.86	0.76
Goeth	2	7	253	0.01	7.39	1.35	1.50	1.23	0.90
Goeth	2	ALL	996	0.00	7.39	0.92	0.96	0.98	1.06
Goeth	3	9	328	0.00	6.42	1.21	1.36	1.17	0.96
Goeth	0	ALL	2,714	0.00	7.39	0.75	1.02	1.01	1.34

Table 14-9: Composite Statistics for Pyrite and Galena

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
FeS	1	1	1	1.50	1.50	1.50	0.00	0.00	0.00
FeS	1	2	86	0.06	6.73	1.28	1.73	1.31	1.02
FeS	1	3	2,647	0.00	10.00	1.26	1.35	1.16	0.92
FeS	1	8	2,786	0.00	12.68	1.29	1.15	1.07	0.83
FeS	1	10	1	2.15	2.15	2.15	0.00	0.00	0.00
FeS	1	ALL	5,521	0.00	12.68	1.28	1.25	1.12	0.88
FeS	2	4	394	0.00	10.21	0.46	0.56	0.75	1.61

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
FeS	2	6	46	0.01	2.63	0.74	0.64	0.80	1.09
FeS	2	7	73	0.00	7.25	0.62	1.17	1.08	1.74
FeS	2	ALL	513	0.00	10.21	0.51	0.66	0.81	1.59
FeS	3	9	197	0.00	17.27	0.75	2.24	1.50	2.01
FeS	0	ALL	7,306	0.00	17.27	1.09	1.25	1.12	1.03
PbS	1	1	1	0.50	0.50	0.50	0.00	0.00	0.00
PbS	1	2	91	0.02	12.27	1.27	3.49	1.87	1.47
PbS	1	3	1,793	0.00	10.00	0.23	0.30	0.55	2.39
PbS	1	8	1,778	0.00	6.34	0.18	0.16	0.39	2.14
PbS	1	10	1	0.22	0.22	0.22	0.00	0.00	0.00
PbS	1	ALL	3,664	0.00	12.27	0.23	0.34	0.58	2.48
PbS	2	4	153	0.00	1.30	0.07	0.02	0.15	2.24
PbS	2	6	32	0.00	0.28	0.07	0.01	0.08	1.09
PbS	2	7	25	0.00	1.00	0.07	0.04	0.20	2.67
PbS	2	ALL	210	0.00	1.30	0.07	0.02	0.15	2.15
PbS	3	9	151	0.00	2.53	0.17	0.10	0.32	1.91
PbS	0	ALL	4,454	0.00	12.27	0.21	0.29	0.54	2.55

Table 14-10: Block Statistics for Silver, Lead, and Zinc

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	1	1	4	58.8	105.7	85.3	411.6	20.3	0.2
Ag	1	2	822	4.5	294.7	81.2	2738.8	52.3	0.6
Ag	1	3	53,120	0.6	521.8	29.4	1005.7	31.7	1.1
Ag	1	8	102,322	0.5	479.6	10.6	203.0	14.2	1.3
Ag	1	10	13	59.4	158.6	103.1	786.8	28.1	0.3
Ag	1	ALL	156,281	0.5	521.8	17.4	590.3	24.3	1.4
Ag	2	4	21,514	0.6	285.4	20.2	641.0	25.3	1.3
Ag	2	6	3,433	0.5	216.0	37.3	887.1	29.8	0.8
Ag	2	7	6,595	0.5	192.1	33.9	527.5	23.0	0.7
Ag	2	ALL	31,542	0.5	285.4	24.9	692.9	26.3	1.1
Ag	3	9	5,901	4.0	308.2	45.8	1368.8	37.0	0.8
Ag	0	ALL	277,500	0.5	521.8	17.1	543.9	23.3	1.4
Pb	1	1	4	1.57	2.49	2.04	0.20	0.45	0.22
Pb	1	2	822	0.33	6.16	2.03	0.68	0.83	0.41
Pb	1	3	51,752	0.00	5.95	0.54	0.24	0.49	0.91
Pb	1	8	101,022	0.00	5.93	0.23	0.08	0.29	1.27
Pb	1	10	13	0.82	2.91	1.85	0.36	0.60	0.33
Pb	1	ALL	153,613	0.00	6.16	0.34	0.18	0.42	1.23
Pb	2	4	21,463	0.01	2.91	0.29	0.10	0.31	1.07
Pb	2	6	3,411	0.01	3.60	0.59	0.29	0.54	0.91
Pb	2	7	6,583	0.01	3.36	0.52	0.15	0.39	0.75
Pb	2	ALL	31,457	0.01	3.60	0.37	0.14	0.38	1.02
Pb	3	9	5,920	0.03	4.98	0.77	0.24	0.49	0.64
Pb	0	ALL	190,990	0.00	6.16	0.36	0.18	0.42	1.17
Zn	1	1	4	0.72	1.54	1.22	0.14	0.37	0.30
Zn	1	2	822	0.05	3.90	0.94	0.32	0.56	0.60
Zn	1	3	52,185	0.01	5.56	0.32	0.20	0.45	1.42
Zn	1	8	100,955	0.01	5.66	0.34	0.10	0.32	0.94

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Zn	1	10	13	0.14	3.06	1.56	1.23	1.11	0.71
Zn	1	ALL	153,979	0.01	5.66	0.34	0.14	0.38	1.11
Zn	2	4	20,712	0.02	1.07	0.11	0.01	0.07	0.69
Zn	2	6	3,402	0.03	1.16	0.15	0.01	0.09	0.60
Zn	2	7	6,561	0.03	0.59	0.12	0.00	0.05	0.45
Zn	2	ALL	30,675	0.02	1.16	0.11	0.01	0.07	0.64
Zn	3	9	5,697	0.01	1.25	0.16	0.01	0.09	0.60

Table 14-11: Block Statistics for Copper, Goethite, and MnOx

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Cu	1	1	4	0.019	0.045	0.035	0.000	0.011	0.321
Cu	1	2	822	0.013	0.175	0.044	0.001	0.026	0.600
Cu	1	3	49958	0.000	0.839	0.028	0.001	0.031	1.109
Cu	1	8	86454	0.000	0.326	0.017	0.001	0.024	1.389
Cu	1	10	13	0.023	0.080	0.049	0.000	0.016	0.328
Cu	1	ALL	137251	0.000	0.839	0.021	0.001	0.027	1.282
Cu	2	4	19025	0.000	0.322	0.015	0.000	0.019	1.226
Cu	2	6	3255	0.001	0.122	0.023	0.000	0.015	0.678
Cu	2	7	6029	0.000	0.083	0.017	0.000	0.010	0.596
Cu	2	ALL	28309	0.000	0.322	0.016	0.000	0.017	1.030
Cu	3	9	5849	0.000	0.159	0.046	0.001	0.023	0.495
Cu	0	ALL	171409	0.000	0.839	0.021	0.001	0.026	1.228
Goethite	1	1	4	0.00	0.02	0.01	0.00	0.01	1.62
Goethite	1	2	520	0.00	0.63	0.07	0.01	0.10	1.31
Goethite	1	3	28423	0.00	2.93	0.11	0.05	0.21	1.92
Goethite	1	8	24314	0.00	3.74	0.06	0.03	0.16	2.93
Goethite	1	10	7	0.00	0.01	0.00	0.00	0.00	1.02
Goethite	1	ALL	53268	0.00	3.74	0.09	0.04	0.19	2.26
Goethite	2	4	19734	0.00	4.31	0.49	0.19	0.43	0.89
Goethite	2	6	3222	0.00	4.06	0.92	0.37	0.61	0.66
Goethite	2	7	6204	0.00	4.49	1.03	0.50	0.71	0.69
Goethite	2	ALL	29160	0.00	4.49	0.65	0.33	0.58	0.89
Goethite	3	9	5492	0.00	5.56	0.72	0.46	0.68	0.94
Goethite	0	ALL	87920	0.00	5.56	0.31	0.24	0.49	1.57
MnOx	1	1	2	0.07	0.33	0.20	0.03	0.19	0.94
MnOx	1	2	755	0.00	2.54	0.34	0.19	0.43	1.26
MnOx	1	3	29351	0.00	4.89	0.29	0.24	0.49	1.69
MnOx	1	8	33184	0.00	2.77	0.18	0.12	0.35	1.97
MnOx	1	10	9	0.05	0.51	0.27	0.02	0.15	0.57
MnOx	1	ALL	63301	0.00	4.89	0.23	0.18	0.43	1.83
MnOx	2	4	14842	0.00	2.29	0.21	0.11	0.34	1.62
MnOx	2	6	2942	0.00	3.61	0.41	0.25	0.50	1.19
MnOx	2	7	5141	0.00	2.47	0.19	0.09	0.29	1.55
MnOx	2	ALL	22925	0.00	3.61	0.23	0.13	0.36	1.56
MnOx	3	9	5510	0.00	3.56	0.77	0.51	0.71	0.92

Table 14-12: Block Statistics for Pyrite and Galena

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Pyrite	1	1	4	0.89	1.29	1.13	0.03	0.17	0.15
Pyrite	1	2	822	0.07	4.84	1.10	0.56	0.75	0.68
Pyrite	1	3	49192	0.00	9.00	1.02	0.67	0.82	0.80
Pyrite	1	8	97831	0.00	10.75	1.09	0.55	0.74	0.68
Pyrite	1	10	13	1.29	4.17	2.94	0.85	0.92	0.31
Pyrite	1	ALL	147862	0.00	10.75	1.06	0.59	0.77	0.72
Pyrite	2	4	17645	0.00	5.13	0.22	0.11	0.34	1.53
Pyrite	2	6	2574	0.00	2.44	0.22	0.15	0.39	1.81
Pyrite	2	7	3736	0.00	2.54	0.14	0.05	0.23	1.68
Pyrite	2	ALL	23955	0.00	5.13	0.21	0.11	0.33	1.60
Pyrite	3	9	5021	0.00	9.94	0.27	0.31	0.56	2.05
Pyrite	0	ALL	176838	0.00	10.75	0.93	0.61	0.78	0.85
Galena	1	1	4	0.24	0.39	0.33	0.01	0.07	0.22
Galena	1	2	822	0.03	6.38	0.95	0.99	0.99	1.05
Galena	1	3	45352	0.00	8.85	0.12	0.08	0.28	2.22
Galena	1	8	82509	0.00	7.70	0.10	0.08	0.28	2.81
Galena	1	10	13	0.13	0.46	0.30	0.01	0.12	0.40
Galena	1	ALL	128700	0.00	8.85	0.11	0.09	0.30	2.61
Galena	2	4	11542	0.00	0.79	0.01	0.00	0.03	2.31
Galena	2	6	1934	0.00	0.23	0.01	0.00	0.03	2.38
Galena	2	7	3022	0.00	0.47	0.01	0.00	0.02	2.63
Galena	2	ALL	16498	0.00	0.79	0.01	0.00	0.03	2.39
Galena	3	9	4706	0.00	1.46	0.06	0.01	0.10	1.62
Galena	0	ALL	149904	0.00	8.85	0.10	0.08	0.28	2.75

14.1.4 Classification

Blocks were coded as measured, indicated, or inferred based on the silver grade estimate. The number of samples and the distance to the nearest sample were the discriminating factors. Silver grades were estimated using a minimum of one composite and maximum of ten composites, with a maximum of three composites from one drill hole. If the number of composite samples is at least 6 and the distance to the nearest composite is less than 12 meters, the block is classified as measured. If the number of composite samples is at least 2, the distance to the nearest composite is less than 50 meters, and the block has not been classified as measured, the block is classified as indicated. All other blocks with an estimated silver grade are classified as inferred.

14.1.5 Metallurgical Recovery Estimate

Prior estimates used mineralization codes that were assigned to the block model based on ore mineralogy and categorizing the metallurgical responses of the deposit. Following a detailed geometallurgical analysis, this approach was modified and replaced with one using a statistical model with zinc grade, elevation, and geologic log data (i.e., pyrite %, galena %, MnO, etc) to create a continuous recovery model/formula for each of the metals. Application of this method resulted in generation of the following formulas for the prediction of metals recoveries.

$$\begin{aligned} Lead\ Recovery = {} & 61.9 - 40.9 \times \max(0, 0.57 - Zn) + 7.7 \times \max(0, galena - 0.38) + 45.4 \\ & \times \max(0, 0.37 - goethite) - 0.12 \times \max(0, elevation - 4891) + 32.9 \\ & \times \max(0, 0.27 - MnOx) - 6.21 \times \max(0, pyrite - 1.07) - 16.4 \times \max(0, 1.07 - pyrite) \end{aligned}$$

Note: Maximum Lead Recovery is 98% and minimum Lead Recovery is 2%. Also note, a small error in the calculation of the elevation component was found in the 2017 FEED, where the block elevation used in the Micromodel software estimate of recovery was one block low (8 meters). This caused a slightly low (~1%) recovery for blocks above the thresholds (4891 and 4901) and was corrected in the 2017 model.

$$\begin{aligned} Zinc\ Recovery &= 78.7 - 50.6\ \times \max(0, 1.02 - Zn) - 0.15\ \times \max(0, elevation - 4901) - 5.4 \\ &\quad \times \max(0, pyrite - 1.9) - 11.2\ \times \max(0, 1.9 - pyrite) + 104.1\ \times \max(0, Cu - 0.03) \\ &\quad + 1620.2\ \times \max(0, 0.03 - Cu) \end{aligned}$$

Note: Maximum Zinc Recovery is 83% and minimum Zinc Recovery is 0%

$$Silver\ Recovery\ in\ Lead\ Concentrate = 0.67 \times Lead\ Recovery + 12$$

$$Silver\ Recovery\ in\ Zinc\ Concentrate = 41 - 0.41 \times (Silver\ Recovery\ in\ Lead\ Concentrate)$$

Figure 14-25 and Figure 14-26 compare predicted silver recovery for the prior and current models along a portion of an E-W cross-section through the block model that includes Corani Minas. Figure 14-27 and Figure 14-28 do the same for zinc. Equivalent comparison sections for lead were presented in Section 7 (Figure 7-9 and Figure 7-10). The cross-sections show how the updated model can capture spatial variability in recovery and provides improved resolution relative to the old model based on discrete geometallurgical categories.

Figure 14-25: Silver Recovery for Previous Model

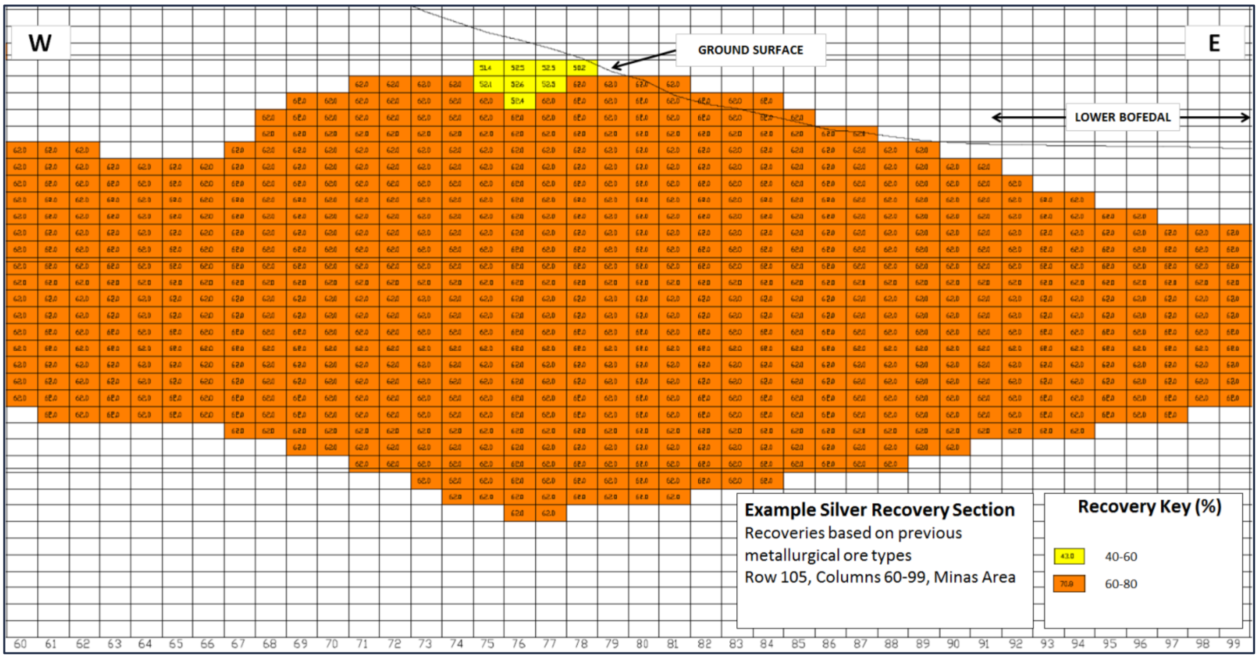


Figure 14-26: Silver Recovery for Current Model

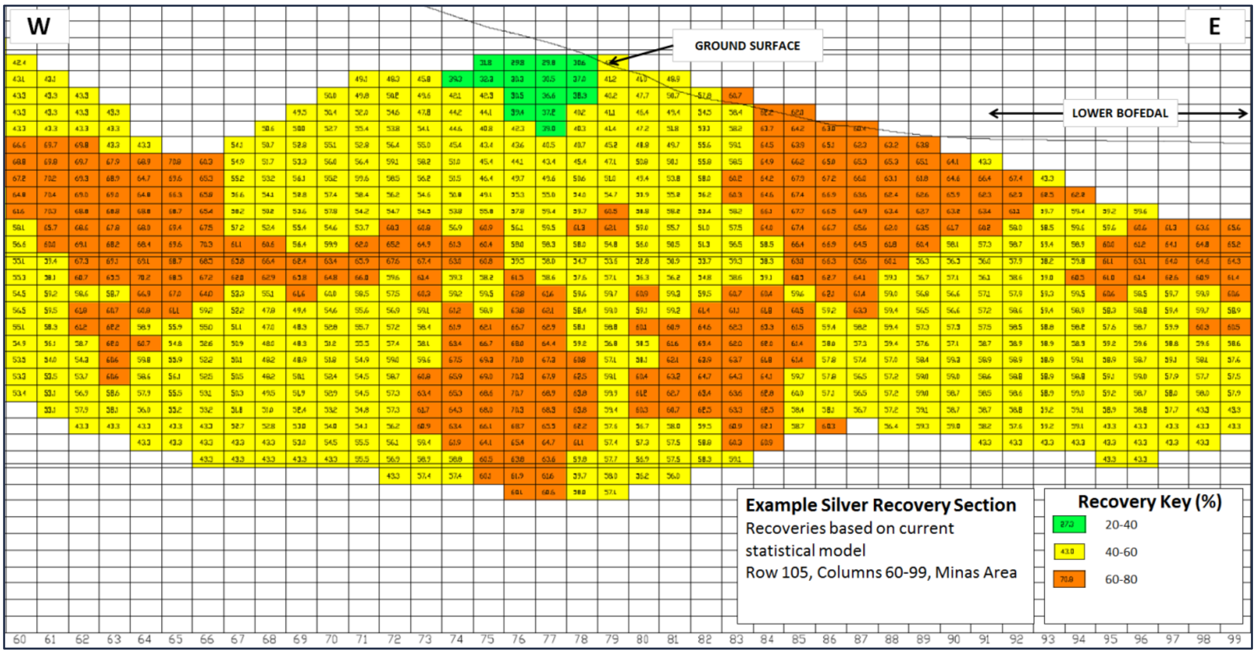


Figure 14-27: Zinc Recovery for Previous Model

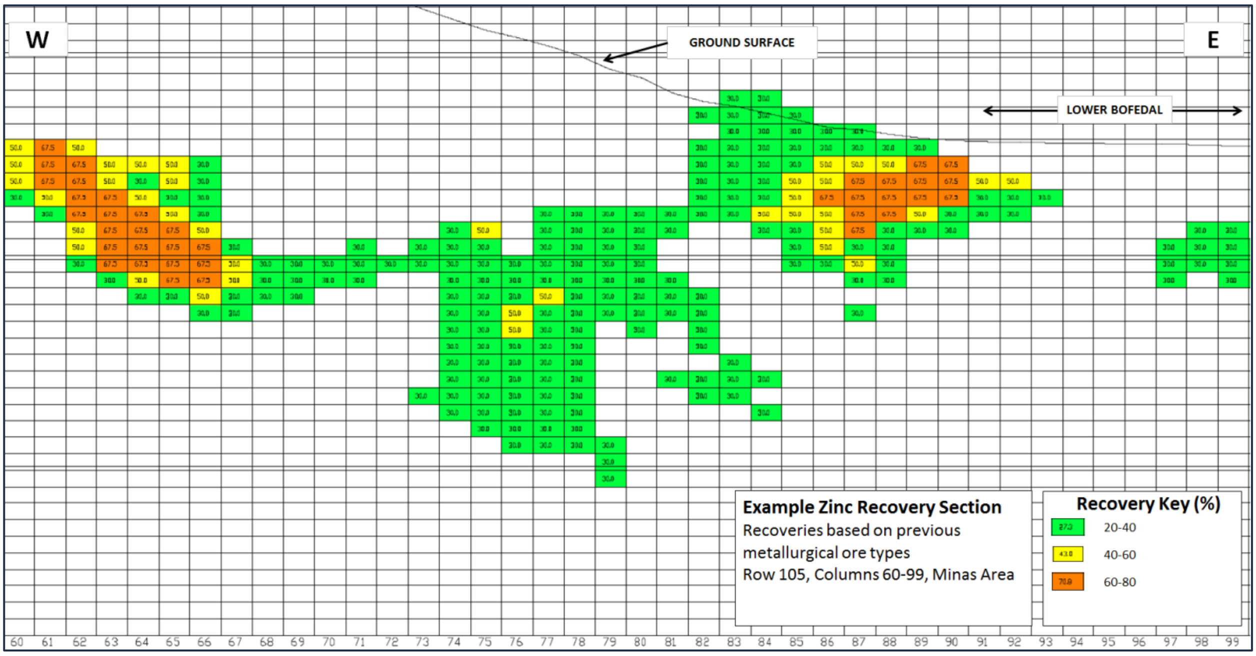
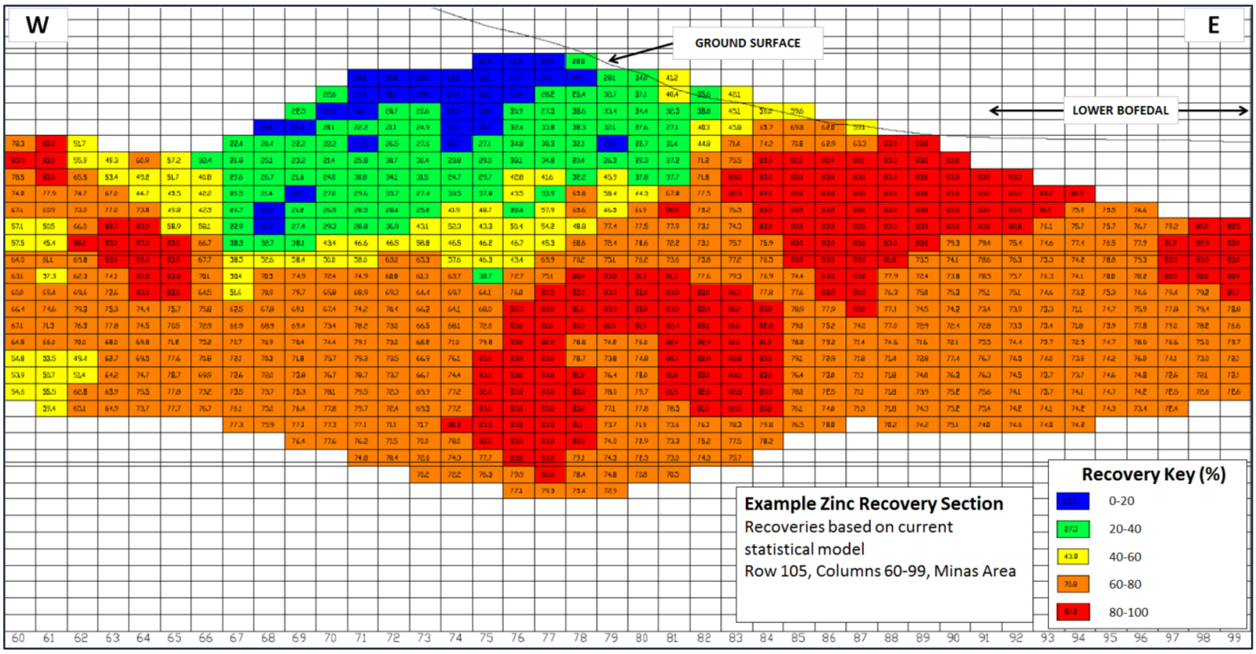


Figure 14-28: Zinc Recovery for Current Model



14.2 Acid Rock Drainage

To use waste rock for construction of the dump, backfill cap, embankment dams, etc., there needed to be a way to distinguish non-acid-generating rock (NAG) from potentially acid-generating rock (PAG). For this purpose, GRE developed an acid rock drainage (ARD) block model using available geochemical sample data collected by Vector in 2009, GRE in 2011, and AMEC in 2012.

Various analyses were performed on each sample group, including full Acid-Base Accounting (ABA) analysis, whole rock analysis, metals by aqua regia digestion, and total carbon/total sulphur assay by LECO furnace. A summary of each sample group and analysis is given in Table 14-13.

Table 14-13: Geochemical Sample Summary

Collected By	Date	No. of Samples	Analyses			
			ABA	LECO	WRA	Metals
Vector	2009	23	23	23	7	7
GRE	2011	224	23**	201	23**	23**
AMEC	2012	397*	397	397	397	397
Total		**644**	**443**	**621**	**427**	**427**
*2 samples are reported as the same location and interval. The results for these samples were averaged **23 Samples from onsite kinetic cell program						

The parameters selected for inclusion in the model included AP/total sulphur, NP, pH, and metals of concern, which are described in the Geochemical Characterization Report (GRE, 2012) and include arsenic, cadmium, copper, lead, mercury, nickel, and zinc.

For the parameters considered, comparable analyses from different labs (i.e., total sulphur/total sulphur/ S-%) were matched, and results were converted to consistent units. GRE developed linear models relating to ABA, AP, and total sulphur by LECO furnace by rock type for samples collected by Vector in 2009 and GRE in 2011. These relationships were used to predict AP for samples where only LECO furnace total sulphur/carbon was analysed. AP was predicted only for samples from rock types PM, FeO, and FBS, for which the relationship between total sulphur and AP had an R^2 greater than 0.97. If the model predicted an AP less than 0, an AP of 0 was assumed. This analysis is described in the Geochemical Characterization Report (GRE, 2012). Table 14-14 and Table 14-15 contain summary statistics for the raw data.

Table 14-14: Raw Data Summary Statistics

Parameter	AP	NP	As	Cd	Cu
Number	610	442	426	426	426
Mean	22.4	3.9	248.6	13.0	94.5
Std Dev	32.7	7.0	518.2	44.1	149.2
Variance	1066.4	48.5	268529.5	1942.5	22265.4
Maximum	206.0	97.0	5560.0	569.0	1200.0
Minimum	0.0	0.00	10.1	0.02	1.5
Range	206.0	97.0	5549.9	569.0	1198.5
Coef Var	145.7	178.0	208.4	339.8	157.9
Std Err	1.3	0.3	25.1	2.1	7.2

Table 14-15: Raw Data Summary Statistics

Parameter	Pb	Ni	Hg	Zn	pH
Number	426	426	426	426	442
Mean	2231.5	8.7	0.9	1518.3	6.2
Std Dev	4412.9	11	1.7	2596.2	1.5
Variance	19473957.0	120.7	3	6740476.8	2.3
Maximum	61900.0	74.8	18.8	32300.0	9.5
Minimum	10.0	0.6	0.01	59.0	2.3
Range	61890.0	74.2	18.8	32241.0	7.2
Coef Var	197.8	126.1	189.7	171.0	24.3
Std Err	213.8	0.5	0.08	125.8	0.07

The geochemical samples were loaded into TECHBASE and composited to 8-meter intervals along lithology boundaries. Inverse distance cubed weighting was used to estimate constituent concentrations in all model blocks contained within a wireframe boundary based on geologic boundaries of the project mineralized bodies. For each block in the block model, 2 to 8 samples were considered within a search ellipsoid, and a weighted average of the parameter of concern was calculated.

Using this model, the spatial variation of acid-generating waste rock within the pit and in the pit walls was determined. Material with an estimated net neutralizing potential less than -20 tonnes/kilotonne was designated as potentially acid generating; material with an estimated net neutralizing potential greater than -20 tonnes/kilotonne was designated as non-acid generating.

14.3 Mineral Resources

The mineral resources were developed with the Whittle multi-algorithm software suite to determine the component of the deposit with reasonable prospects of economic extraction. For the resource pit shell, economic benefit was applied to inferred mineralization. However, no economic benefit was applied to inferred mineralization in the later determination of mineral reserves or in the economic analysis of the project.

The Whittle computer algorithm is a tool used to guide mine design. The algorithm applies approximate costs and recoveries along with approximate pit slope angles to establish theoretical economic breakeven pit wall locations that result in a maximum pit value.

The Whittle algorithm was also used as a guide to the design of the final mineral reserve pit. The detailed cost and process response input parameters for the mineral reserves are presented in Section 15. The majority of that information was also used as input for the determination of mineral resources.

The economic input for mineral resource determination was identical to that applied to the mineral reserve, with the following exceptions:

1) The resource Whittle pit shell did receive economic credit for inferred-class material. Inferred was treated as waste for the mineral reserve.

2) The Mineral Resources were generated within the $30.00 silver, $1.425 lead, and $1.50 zinc price pit shell and the calculated $11/tonne NSR cut-off (see Table 14-16).

3) The Mineral Resource contains potentially leachable material processed at $4.82/tonne and above a 15 g/tonne silver cut-off (Table 14-17). This Resource is contained within the Whittle pit shell but is not included in Table 14-16. The Mineral Reserve does not include any potentially leachable material.

The resource statement on Table 14-16 includes the mineral reserves that are presented in Section 15.

The qualified person responsible for the estimation of the Mineral Resource was Terre Lane. Metal price changes could materially change the estimated mineral resources in either a positive or negative way.

At this time, there are no unique situations relative to environmental, permitting, legal, title, taxation, political, or socioeconomic conditions that would put the Corani mineral resource at a higher level of risk than any other Peruvian developing resource.

Table 14-16: Total Mineral Resources (Includes Both Resources and Reserves)

Category	Ktonnes	Silver gpt	Lead %	Zinc %	Silver Million oz	Lead Million lb	Zinc Million lb
Measured	29,209	56.2	0.912	0.582	52.8	587	375
Indicated	181,902	40.7	0.741	0.495	238	2971.3	1983.5
Measured + Indicated	211,111	42.8	0.765	0.507	291	3,558	2,359
Inferred	31,231	40.6	0.742	0.512	40.8	510.6	352.4

Note: Cut-off Value: $11.00/tonne covers process and general and administrative costs.

Table 14-17: Total Mineral Resource of Potentially Leachable Material (Includes the Mineral Reserve)

Category	Ktonnes	Silver gpt	Silver Million oz
Measured	5,006	38.0	6.12
Indicated	19,690	23.1	14.61
Measured + Indicated	24,697	26.1	20.72
Inferred	8,722	25.1	7.03

Figure 14-29, Figure 14-30, and Figure 14-31 compare cumulative frequencies of silver, lead, and zinc. Each plot shows the cumulative frequency of composite grades, nearest-neighbour block grades, inverse distance block grades, and kriged block grades. The plots show how a nearest-neighbour model estimate follows the trends of the composite assay data very closely in all of the metal models. The inverse distance and kriged models show how, once spatial variability is applied, the frequency plot of the population takes on the smoothed distribution that is expected of a grade model.

Figure 14-32 and Figure 14-33 are examples of a plan and cross-section of the block model with drill hole intercepts plotted with composite assay data and the block grade model for silver. The visual inspection shows how the model agrees with the assay data and smooths the grade among input data points.

Figure 14-29: Cumulative Frequency of Silver Grades – Composites, Kriged, ID3, and Nearest Neighbour

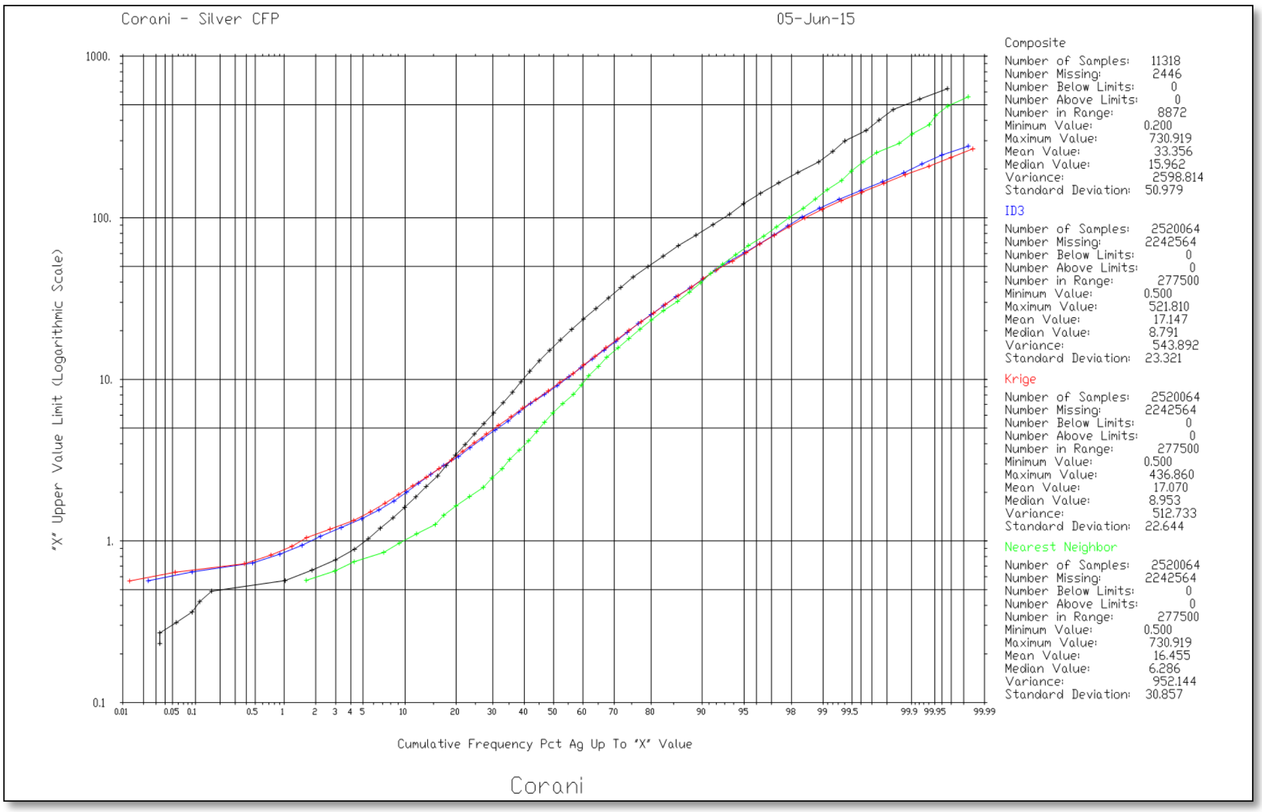


Figure 14-30: Cumulative Frequency of Lead Grades – Composites, Kriged, ID2.5, and Nearest Neighbor

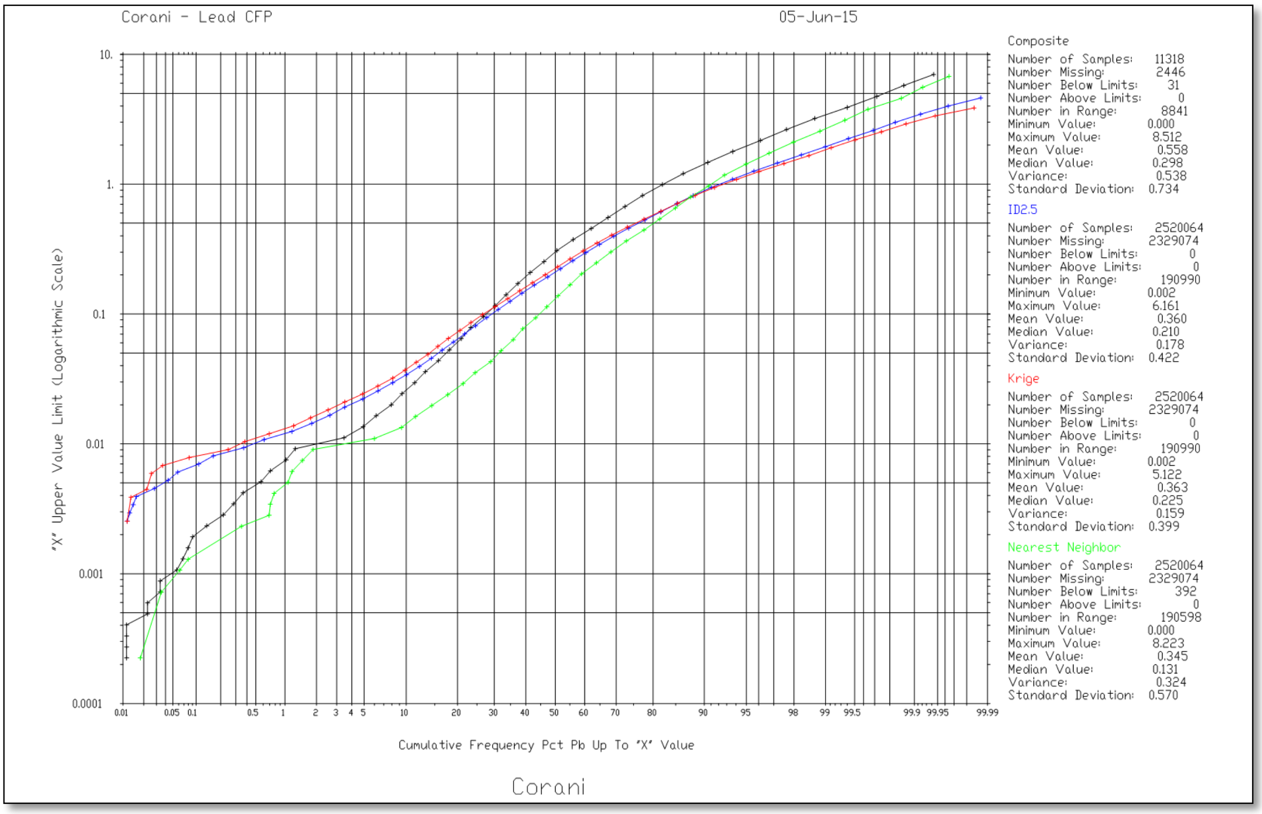


Figure 14-31: Cumulative Frequency of Zinc Grades – Composites, Kriged, ID2.5, and Nearest Neighbor

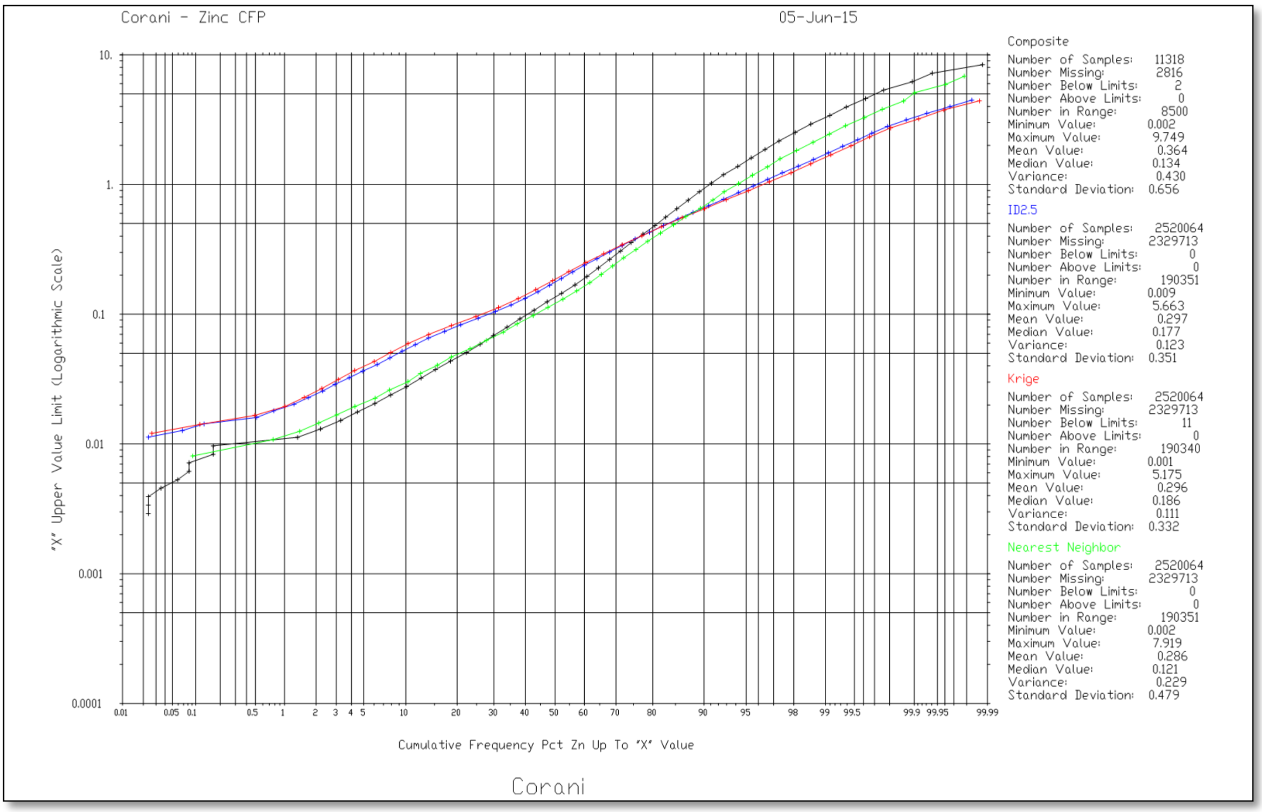


Figure 14-32: Silver Composite Assays and ID3 Block Grades on Bench 4914

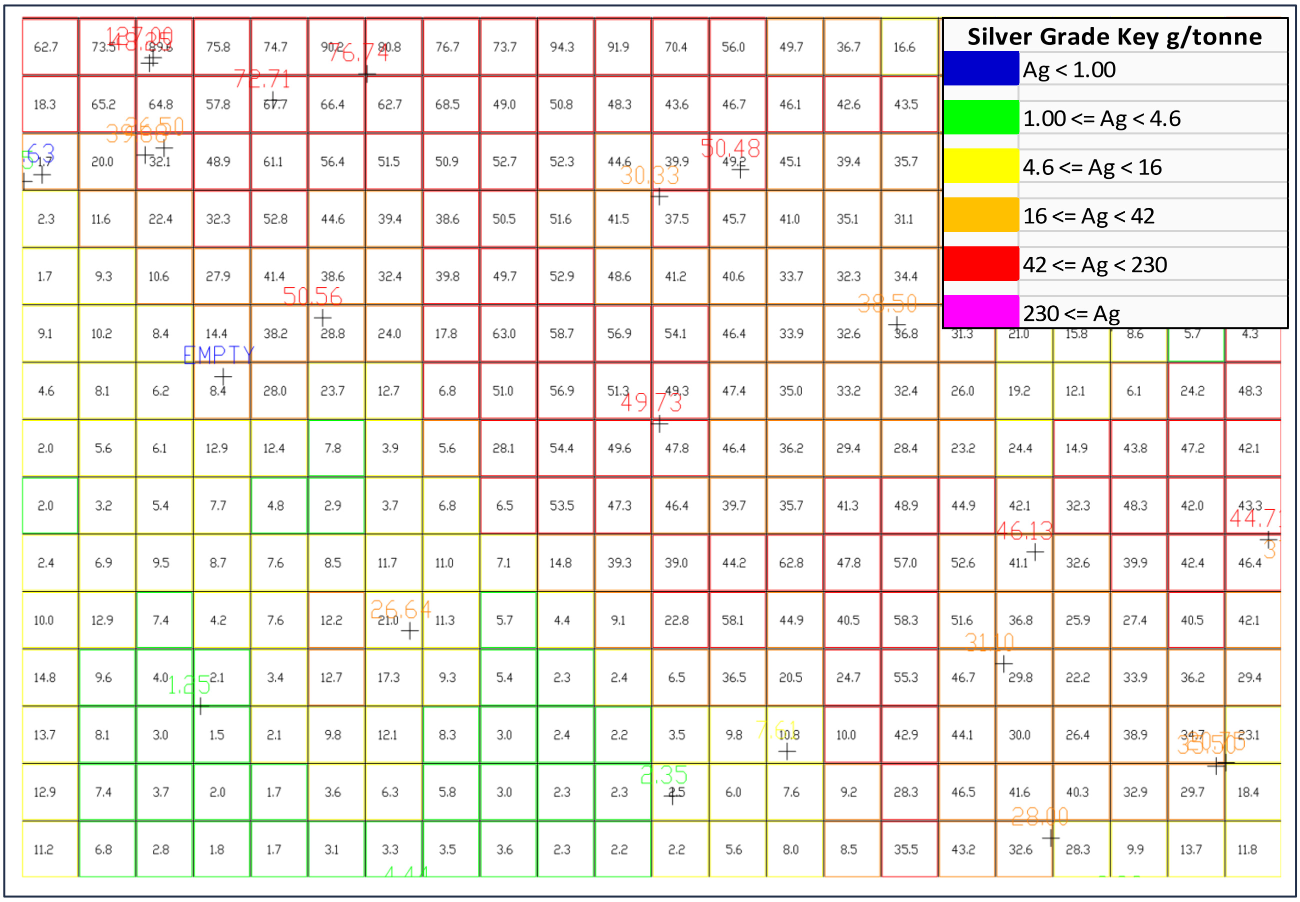
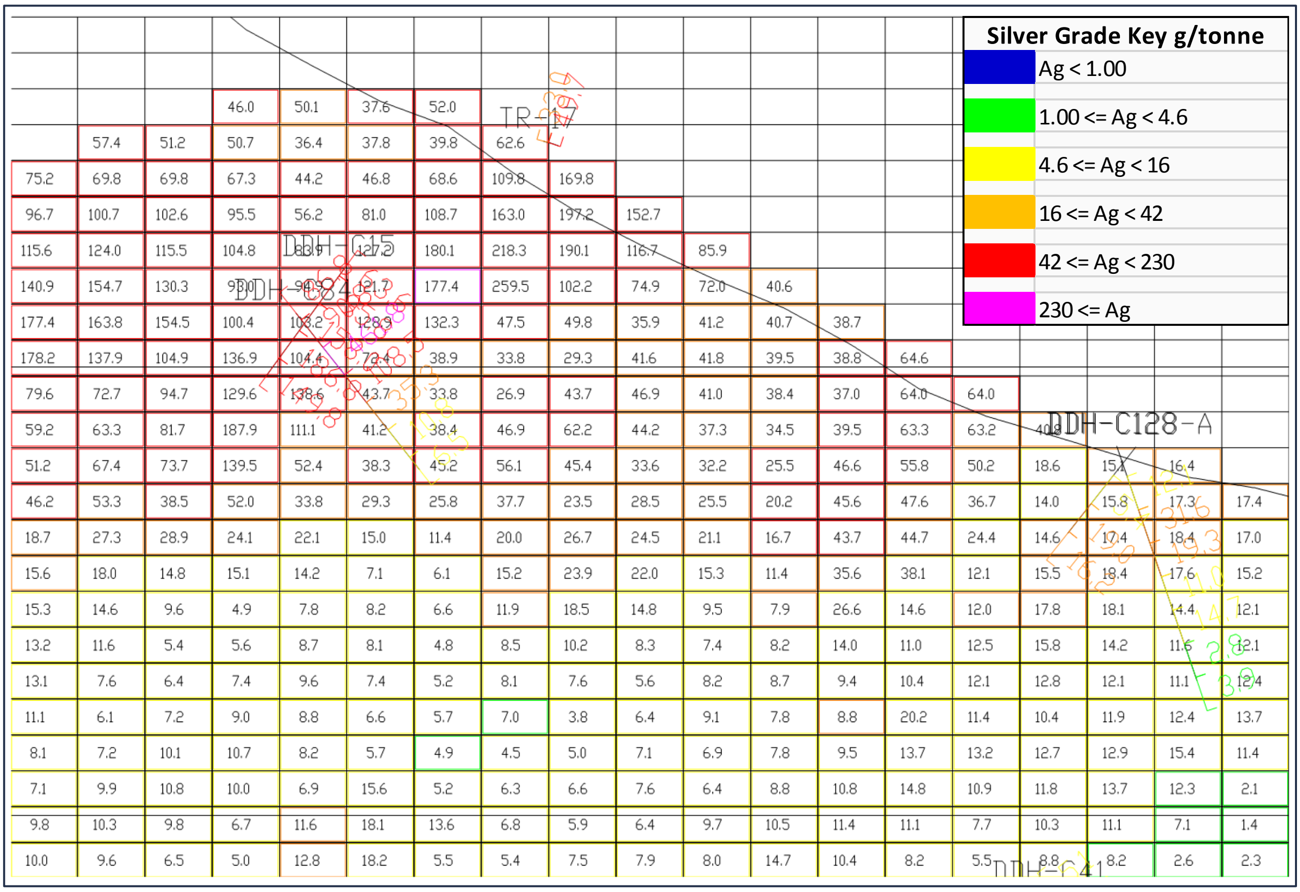


Figure 14-33: Silver Composite Assays and ID3 Block Grades Cross-section 8,448,252.5 N



15 Mineral Reserve Estimates

GRE has reviewed and verified that the mine design, Mineral Reserve Estimate, and production schedule generated by GMI and GyM have been prepared with sound engineering principles and are correct. The work uses the Mineral Resource block model created by GRE in 2015 as a basis for the work. GRE has found the work performed by GMI/GyM for the Detailed Engineering Phase 1 (FEED) to be well done and thorough, and the Mineral Reserves stated herein represent a slightly improved mine plan over the GRE 2015 Feasibility Study Estimate. GMI/GyM used \$20.00/oz silver, \$1.00/lb zinc, and \$0.95/lb lead for the mine design, and economic model base case metal prices of \$18.00/oz silver, \$1.10/lb zinc, and \$0.95/lb lead. The mine plan also uses a variable cut-off value higher than the economic cut-off value (these cut-offs would be equivalent to the economic cut-off calculated at significantly higher metal prices). This combined approach utilizing higher metal prices for the pit design and lower metal prices in the economic model is excellent for projects nearing production in a potential rising metal price environment, as it is can be difficult or impossible to make small changes to the pit design and size after mining begins because of bench width and equipment maneuvering requirements. The elevated cut-off increases cash flow, reduces risk, and improves the economic performance of the project.

The following text describes the work done by GMI/GyM.

The mineral reserve is the total of all proven- and probable-category ore that is planned for production. The mine design and mine plan presented in Section 16 details the production of that reserve. The mineral reserve was established by tabulating the contained tonnes of measured and indicated material (proven and probable) within the designed final pit geometry that are fed to the process plant in the economic model. The final pit design and the internal phase designs were guided by the results of Whittle pit optimization analyses.

15.1 Introduction

As part of the Front End Engineering Design being undertaken by GMI for Bear Creek Mining´s Corani Project, GyM has carried out the mining engineering design aspects of the project. The mine, located between 4,800 and 5,000 m above sea level in the Andes mountains of Peru, will be an open pit polymetallic operation, producing Pb/Ag and Zn Concentrates.

GyM has used the geological block model from the 2015 Feasibility Study and re-optimised the project, using updated parameters. A major change compared to the 2015 Feasibility Study is the decision to not mine the upper and lower bofedals and the natural wall of rock which separate them. This has resulted in two separate pits (East and Minas/Main) separated by the bofedals, which removes operational risk associated with mining of bofedals and will make the overall project simpler.

Based on the updated mine design, a mine schedule and process plant feed schedule has been developed to maximize the project value. Capital, Sustaining Capital, and Operating costs have been estimated based on the mine schedule, for owner operator and contractor scenarios.

15.2 Pit Optimization

15.2.1 Block Model

The resource block model used was supplied by Bear Creek Mining, and is called GAP Solutions_V2.csv (file size 560.387Mb). The coordinate system of the resource block model is PSAD56. The block model consists of blocks which measure 15 m (north - south) x 15 m (east – west) x 8 m (vertical).

It was not within the scope of the FEED study to review the model from a geological or resource classification perspective.

For the FEED study, the resource block model was modified slightly, creating the model used in the FEED Corani_geo162612_B.csv. The purpose of modifying the model was to correct some minor inconsistencies and to create fields that were required as part of the FEED. The modifications are documented in reports 161655-050-15-INF-0001_Rev 0 (Preparación de Modelo Geológico) and 161655-050-15-INF-0002_Rev 0 (Preparación de Modelo Planeamiento).

The modified resource block model, Corani_geo162612_B.csv, has the fields and descriptions shown in Table 15-1.

Table 15-1: Modified Resource Block Model Fields

Field	Description	Units
EASTt	Block Centroid Easting	m
NORTH	Block Centroid Northing	m
ELEV	Block Centroid Elevation	m
AG	Silver grade	g/t
PB	Lead grade	%
ZN	Zinc grade	%
CU	Copper grade	%
GOET	Goethite grade	%
MNOX	Manganese oxide grade	%
GALE	Galena grade	%
PYRT	Pyrite grade	%
SG	Specific gravity	t/m^3
CLASS	Classification based on GRE's Model – 1 = measured, 2 = indicated, 3 = inferred	
BOFED	Indicator: whether block is on bofedal – 1 = yes, 0 = no	
LITO	Lithologic unit: 1 = cover 1; 11 = post-mineral tuff; 20 = pre-mineral tuff; 31 = sediments	
MINER	Mineral Domains: 1 = CS, 2 = CSC, 3 = FBS, 4 = FEOX, 6 = MNO, 7 = PG, 8 = PM, 9 = QSB, 10 = TET	
PTOPO	Topo percent (0-100)	
ZONE	Zone code: 1 = Main, 2 = Minas, 3 = Este, 4 = Bofedal	
NNP	Net neutralization potential. It is the difference between NP and AP and can be indicative of the potential for a material to produce ARD. An NNP value of tonnes/kilotonne was used as a cut-off value: blocks with NNP values less than -20 are designated as PAG, and blocks with values greater than -20 are designated as NAG (non-acid generating)	t/kt
PAG	Indicator: acid generating block. 1 = PAG, 0 = NAG	

Resource Block Model Corani_geo162612_B.csv, used for the FEED.

15.2.2 Topography

The topographic surface used with the bock model and for optimization and mine design consisted of the following files. These files are from the 2015 Feasibility Study and are in the WGS84 system. For the FEED, they were converted to PSAD56 (using ArcGIS software) for use with the block model.

- corani pit area topo – trimmed by MES and cloud fixed.dxf
- corani_1m_contours_north.dwg
- corani_1m_contours_south.dwg
- corani_5m_contours.dwg

- corani_10m_contours.dwg
- corani_50m_contours.dwg

Once the mine design was competed, all designs were converted back to the WGS84 system for incorporation into the overall project.

It is recommended that before construction commences, Bear Creek Mining convert the block model to the WGS84 system. This may require surveying of exploration drill hole collars in the field to determine their coordinates in the WGS84 system and so should be done prior to construction as some hole collars are likely to get destroyed.

15.2.3 Optimisation Parameters

Pit optimisations were undertaken using Whittle software (Lerch Grossman algorithm), and SimSched (a direct block scheduling software) to identify final pit limits. Following design of the final pit, and internal phases, Comet software was used to determine the mining sequence which maximizes the project NPV when taking into account physical constraints applied. For all three processes, the following revenue, processing, cost, and geotechnical parameters detailed in report 161655-2300-15-CD-001 Rev0 (Mine Optimisation and Planning Input) were used. Only a brief summary is given here, and the reader is directed to the relevant report for more detail.

15.2.3.1 Revenue Parameters

All revenue factors have been taken from the report “Optimized and Final Feasibility Study” (M3, 30 May 2015). Table 15-2 shows the metal prices used.

Table 15-2: Revenue Factors

Metal	Unit	Reserves
Lead	$/lb	0.95
Zinc	$/lb	1.00
Silver	$/oz	20.00

The Detailed Design Phase 1 (FEED) economic model uses slightly lower metal prices of $18.00/oz silver, $1.10/lb zinc, and $0.95 lead and uses cut-off grades higher than the economic cut-off.

Selling costs have been taken from the report “Marketing and Sales Cost Study on Corani Pb/Ag and Zn Concentrates” (Andes Mining Research, Dec 2014), provided by Bear Creek Mining, and include the treatment charges and refining charges shown in Table 15-3.

Table 15-3: Treatment and Refining Charges

Description	Unit	Pb/Ag Conc.	Zn Conc.
Treatment Charge (TC)	$/t(dry)	208.00	208.00
Refining Charge (RC)	$/oz Ag Payable	1.50	0.00

Other costs associated with selling, freight, penalties, and royalties / taxes are detailed in report 161655-2300-15-CD-001 Rev0 (Mine Optimisation and Planning Input).

15.2.3.2 Processing Parameters

The processing plant throughput used was 7.875 Mt (dry) per year. The project is considered to be “processing limited” in Whittle, rather than being limited by mining or selling capacity.

For optimization work, blocks coded as Post-mineralization and Pre-mineralization were considered for processing, whilst blocks coded as Cover or Basement Sediments were treated as waste. Note: in the final mine schedule and processing schedule, the Post-mineralization material was considered as waste. The quantity of this material that was reclassified from ore to waste was on the order of 2.1 Mt and is considered minor in relation to the overall project.

The following formulae, sourced from the report "Optimized and Final Feasibility Study" (M3, 30 May 2015) were used for calculating recoveries to the Pb-Ag and Zn concentrates.

$$Lead\ Recovery = 61.9 - 40.9 \times \max(0, 0.57 - Zn) + 7.7 \times \max(0, galena - 0.38) + 45.4 \times \max(0, 0.37 - goethite) - 0.12 \times \max(0, elevation - 4891) + 32.9 \times \max(0, 0.27 - MnOx) - 6.21 \times \max(0, pyrite - 1.07) - 16.4 \times \max(0, 1.07 - pyrite)$$

Note: Maximum Lead recovery was capped at 98%, and any recoveries less than 2% were considered to be 0%.

$$Silver\ Recovery\ in\ Lead\ Concentrate = 0.67 \times Lead\ Recovery + 12$$

$$Zinc\ Recovery = 78.7 - 50.6 \times \max(0, 1.02 - Zn) - 0.15 \times \max(0, elevation - 4901) - 5.4 \times \max(0, pyrite - 1.9) - 11.2 \times \max(0, 1.9 - pyrite) + 104.1 \times \max(0, Cu - 0.03) + 1620.2 \times \max(0, 0.03 - Cu)$$

Note: Maximum Zinc recovery was capped at 83%, and any recoveries less than 0% were considered to be 0%.

$$Silver\ Recovery\ in\ Zinc\ Concentrate = 41 - 0.41 \times (Silver\ Recovery\ in\ Lead\ Concentrate)$$

Note that in the case of silver, for the final concentrate production schedule, which was developed following the optimization, mine design, and mine schedule stages, the total silver recovered (according to the formulae above) was redistributed differently between the Pb-Ag and Zn concentrates, based on lock cycle test results documented in the report "Corani Geomet Block Model Resource ISSUED Sept 18 2015." The Ag grade in the Zn concentrate was lowered from 1,337 g/t to 385 g/t, with the difference being transferred to the Pb-Ag concentrate. This adjustment is consistent with that used in the report "Optimized and Final Feasibility Study" (M3, 30 May 2015).

15.2.3.3 Cost Parameters

Mining Costs

Table 15-4 shows the mining costs used as input to the optimization process for material other than bofedal. The mining cost input used for bofedal material was $6.38/t(dry), consistent with the 2015 feasibility study.

Table 15-4: Mining Costs (Other than Bofedal)

Pit Area	Unit	East Ore	East Waste	Main & Minas Ore	Main & Minas Waste
Reference Bench (mid-level)	mRL	4,846	4,846	4,894	4,894
Drilling	$/t(dry)	0.09	0.09	0.09	0.09
Blasting	$/t(dry)	0.17	0.17	0.17	0.17
Loading	$/t(dry)	0.22	0.22	0.22	0.22
Hauling	$/t(dry)	0.18	0.62	0.19	0.48
Ancillary	$/t(dry)	0.18	0.18	0.18	0.18
Dewatering	$/t(dry)	0.02	0.02	0.02	0.02
Labor – Operators	$/t(dry)	0.11	0.11	0.11	0.11
Labor – Mechanics	$/t(dry)	0.05	0.05	0.05	0.05
Mine Supervision	$/t(dry)	0.10	0.10	0.10	0.10
Maintenance Supervision	$/t(dry)	0.08	0.08	0.08	0.08

Mine Planning, Survey, Geotech	$/t(dry)	0.03	0.03	0.03	0.03
Geology	$/t(dry)	0.04	0.04	0.04	0.04
Geology – Assays	$/t(dry)	0.04	0.04	0.04	0.04
Total at Reference Bench	$/t(dry)	1.308	1.748	1.32	1.61
Incremental Haulage Cost	$/t(dry)/m above	0.00113	0.00113	0.00113	0.00113
Incremental Haulage Cost	$/t(dry)/m below	0.00175	0.00175	0.00175	0.00175

Following the end of mining at Main and Minas Pits, these pits must be backfilled (with waste sourced from the Main Waste Dump) to approximately 4,855 masl as stated in the Feasibility Study. Any block mined below this level at these pits creates a future backfilling cost. For optimizations, a further mining cost of $1.00/t (dry) was added to blocks below this level to reflect the additional backfill cost generated.

Processing Costs

The processing costs are shown in Table 15-5.

Table 15-5: Processing Costs

Description	$/t(dry)
Processing Operating Cost (Direct and Indirect Cost) Processing	8.76
Sustaining Capex Cost	0.15
Tailings Transport and Placement Cost	1.09
Total	10.00

The input value for the Processing Operating Cost and the Sustaining Capex Cost come from the Feasibility Study. The Tailings Transport and Placement Cost is a new cost that has been added as part of the FEED, and the input cost was estimated to be $1.09/t (dry) processed by calculating typical haul cycles and productivities to the centroid of the Main Mine Waste and Filtered Tailings Deposit, and applying equipment costs. An Ore Premium cost was subtracted from the processing cost of $10.00/t (dry) as the cost of hauling material to the process plant crusher is less than hauling it to the waste dump. A general and administration cost of $1.55/t (dry) processed was added to the processing cost as the project is considered “processing limited.”

15.2.3.4 Geotechnical Parameters

The McDonald Engineering report (Open Pit Geotechnical Study – Draft – June 2012) was used as the basis for pit optimization and design.

The overall and inter-ramp angles (IRA) used are shown in the table below, taken from Table 8.1 of the McDonald report.

As McDonald did not mention the bofedals that may intersect the final pit limits (upper, lower, and several other smaller bofedals), 15 degrees was used, the same as in the “Optimized and Final Feasibility Study” (M3, 30 May 2015). Table 15-6 shows the slope used.

Table 15-6: Slope Angles

Domain	Bench Face Angle degree	Operating Bench Height m	Operating Inter-Ramp Angle degree	Overall Slope Angle degree
Este Pre-Mineral Tuff	70	8	46	46
Este Post-Mineral Tuff	70	8	46	46
Principal Pre-Mineral Tuff	70	8	46	45
Minas Pre-Mineral Tuff	65	8	42	42
Minas Post-Mineral Tuff	65	8	42	42

15.2.4 Whittle Optimization Results

For each block in the resource model, the Net Smelter Return (NSR) was calculated. Net Smelter Return is the net revenue less freight, smelting, and refining costs and deductions.

Optimizations were undertaken in Whittle software for 41 different cases to determine sensitivities to resource classification, metal prices, types of lithology processed, mining costs, processing + G&A costs, recoveries, pit wall angles, and not mining of the principal bofedals.

The Whittle input parameters and results are shown in Table 15-7 and are compared with the parameters and results from the 2015 Feasibility Study. In this comparison, the upper and lower bofedals are mined in both cases, and results shown are for the revenue factor 1.00 shell. The FEED results are from Whittle run 3.

Table 15-7: Whittle Input Parameters and Results

Parameter	Unit	NI43-101 (31 May 2015)			FEED		
		East	Main	Minas	East	Main	Minas
Metal Price – Ag	$/oz	20.00	20.00	20.00	20.00	20.00	20.00
Metal Price – Pb	$/lb	0.95	0.95	0.95	0.95	0.95	0.95
Metal Price – Zn	$/lb	1.00	1.00	1.00	1.00	1.00	1.00
Wall Angles (Overall): Bofedal	degrees	15	15	15	15	15	15
Wall Angles (Overall): Post-Mineralization	degress	46	46	46	46	45	42
Wall Angles (Overall): Pre-Mineralization	degress	42	42	42	46	45	42
Mining Dilution	%	0	0	0	0	0	0
Mining Recovery	%	100	100	100	100	100	100
Mineral Types Processed	Code	?	?	?	1,2,3,4,6, 7,8,9,10	1,2,3,4,6, 7,8,9,10	1,2,3,4,6, 7,8,9,10
Lithology Types Processed	Code	?	?	?	Post(11), Pre(20)	Post(11), Pre(20)	Post(11), Pre(20)
Resource Classifications for Ore	Code	M+I	M+I	M+I	M+I	M+I	M+I
Processing Cost	$/t(dry) Proc	9.37	9.37	9.37	8.76	8.76	8.76
Processing Sustaining Capex Cost	$/t(dry) Proc	0.11	0.11	0.11	0.15	0.15	0.15
G&A Cost	$/t(dry) Proc	1.51	1.51	1.51	1.55	1.55	1.55
Tailings Transport	$/t(dry) Proc	0.00	0.00	0.00	1.09	1.09	1.09
Ore Premium Cost	$/t(dry) Proc	0.00	0.00	0.00	-0.44	-0.29	-0.29
Average Processing Cost (Whittle)	$/t(dry) Proc	10.99	10.99	10.99	11.11	11.26	11.26
Average Mining Cost (Whittle)	$/t(dry) Mined	1.60	1.60	1.60	1.86	1.86	1.86
Ore	Mt (dry)	190..0			157.1		
Waste	Mt (dry)	216.1			204.3		

Parameter	Unit	NI43-101 (31 May 2015) East	Main	Minas	FEED East	Main	Minas
Total	Mt (dry)		406.1			361.4	
Strip Ratio	w:o		1.14			1.30	
Process Throughput	t (dry)/year		7,875,000			7,875,000	
Discount Rate	%		5.0			5.0	
Ag Grade	g/t		44.7			49.6	
Pb Grade	%		0.81			0.88	
Zn Grade	%		0.53			0.55	
Ag In Situ	Moz		273			251	
Pb In Situ	kt		1,544			1,375	
Zn In Situ	kt		1,003			864	
Payable Ag*	%		168			151	
Payable Pb*	%		866			791	
Payable Zn*	%		552			489	
Ag Avg. Payable Recovery*	%		61.5			60.2	
Pb Avg. Payable Recovery*	%		56.1			57.6	
Zn Avg. Payable Recovery*	%		55.1			56.6	
Average NSR	$/t(dry) Proc		27.49			28.91	
Margin (UDCF)	$/t(dry) Proc		12.76			13.43	
DCF (Best)	$/t(dry) Proc		9.48			10.28	
Total UDCF (Best)	M$		2,425			2,110	
Total DCF (Best)	M$		1,802			1,615	

* Recovered grades included the mill recovery and the smelter tems so payable ounces and recoveries are reported.

Comparing the FEED optimization results for Case 3, at a 1.00 revenue factor, with the comparable shell generated during the Feasibility Study, Case 3 has 32.9 Mt (17%) less ore, and the metal content is 10% less on average. The undiscounted cash flow of the FEED Case 3 shell $2,110 M and is $315 M less than the Feasibility Study of $2,425 M. This difference is primarily due to including the following relevant cost items in the optimization which were not included in the feasibility study optimizations:

- Filtered tailings transport M$171
- Mining Fleet Capital and sustaining capital M$80
- Backfill of Minas and Main pits M$29
- Updated processing and G&A costs M$34
- Total M$314

Table 15-8 shows Whittle results for two further scenarios evaluated where the upper bofedal was not allowed to be mined and where both the upper and lower bofedals were not allowed to be mined, compared to the May 2015 Feasibility Study and the Base Case of the FEED study. The table shows that if the upper and lower bofedals are not mined, the undiscounted cashflow (based on the revenue factor 1.00 shells) would lower from $2,110 M to $2,048 M (3%).

Table 15-8: Whittle Results for Additional Cases

Description	Unit	FS 2015 Mine Bofedals	Case 3 Mine Bofedals	Case 75 Dont Mine Upper Bofedal	Case 73 Don't Mine Upper or Lower Bofedals
Ore	t(dry)	190.0	157.1	152.3	145.9
Waste	t(dry)	216.1	204.3	192.8	186.9
Total	t(dry)	406.1	361.4	345.1	332.8

Strip Ratio	w:o	1.14	1.30	1.27	1.28
Ag In Situ	g/t	44.7	49.6	49.7	50.4
Pb In Situ	%	0.81	0.88	0.88	0.89
Zn In Situ	%	0.53	0.55	0.56	0.57
Ag Contained	Moz	273	251	243	236
Pb Contained	kt	1,544	1,375	1,341	1,299
Zn Contained	kt	1,003	864	857	837
Cash flow (Undiscounted)	M$	2,425	2,110	2,078	2,048

In parallel, optimizations were also run in SimSched software, which is a direct block scheduler that aims to maximize the NPV of a project. Results for the case where the upper and lower bofedals are not mined were similar to the corresponding Whittle run 73 with a revenue factor of 0.94.

The whittle optimization and SimSched results are presented in detail in report 161655-050-15-INF-0003_Rev 0 (Construcción del Modelo Whittle).

15.2.5 Selection of Final Pit Limit

After reviewing the optimization results together with Bear Creek Mining, it was decided jointly to advance the project based on a pit design that does not mine the upper or lower bofedals at this stage, or the natural rock wall which separates them. The corresponding run in Whitttle is run 73. The key reasons for this decision were:

- Not having pit walls located in bodedal material would eliminate the associated geotechnical risks
- Eliminate operational and cost risks associated with mining of bofedal material
- The reserves already published by Bear Creek Mining can be maintained without mining the bofedals
- The bofedals and the ground beneath them could still be mined in the future if required.

The Whittle shell selected was the revenue factor 1.00 shell, containing 145.9 Mt of ore above the economic break-even cut-off grade. This shell was selected to allow the use of higher cut-off grades in early periods and to give flexibility so the Zn/Pb ratio can be maintained above 0.4 whenever possible, whilst maintaining the currently published reserve of 137.7 Mt of ore (NI 43-101 report, M3 Engineering, 2015).

Figure 15-1 shows the Whittle shell (run 73, revenue factor 1.00) selected as the basis for the mine design.

Figure 15-1: Whittle Run 73 Shell

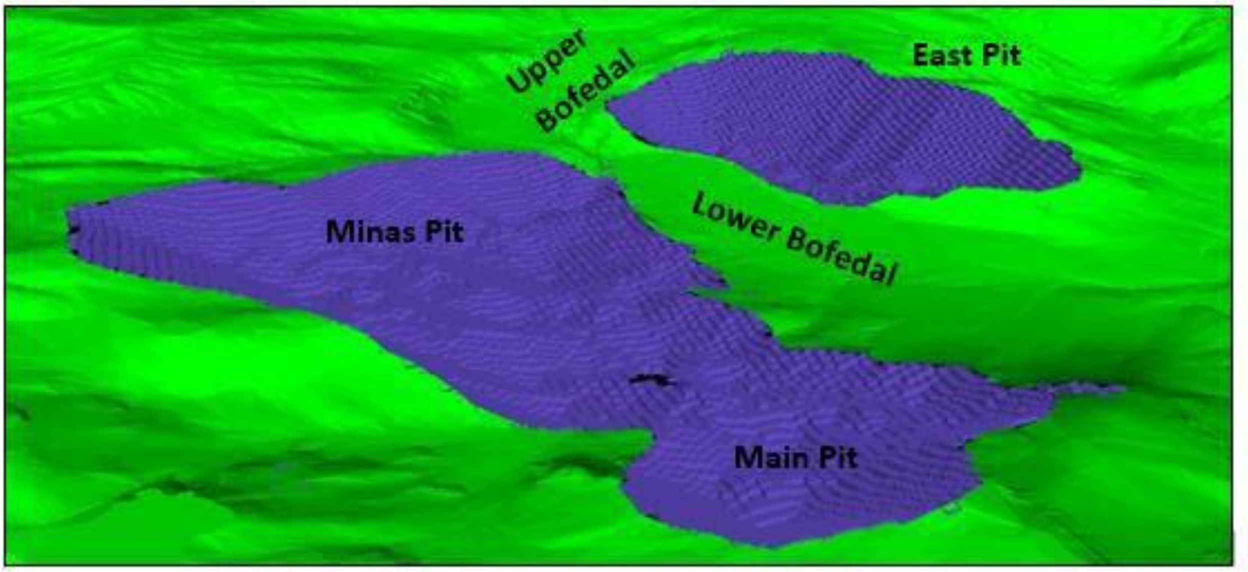
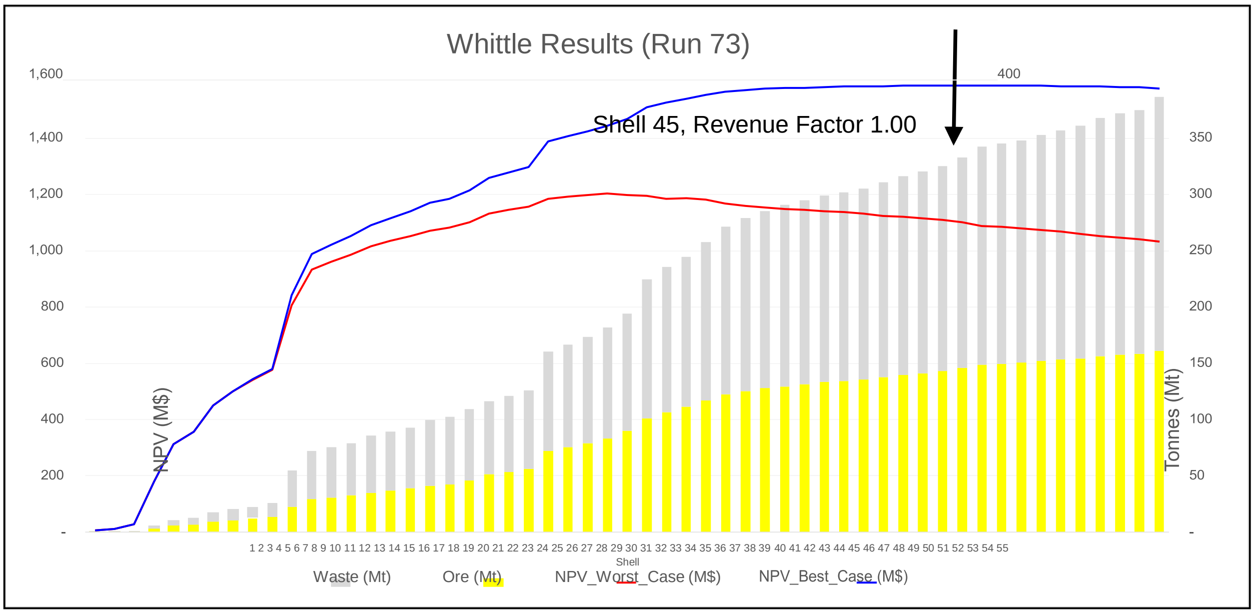


Figure 15-2 shows the Whittle results for run 73. Shell 45 corresponds to revenue factor 1.00.

Figure 15-2: Whittle Run 73 Results



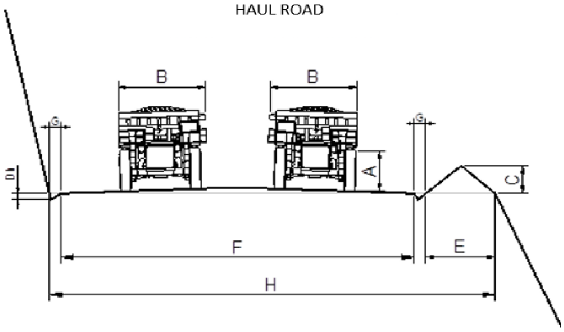
15.3 Mine Design

15.3.1 Final Pit Design

The final pit was designed based on the use of 135-t capacity haul trucks, with ramp widths of 29 m in accordance with width requirements of the Peruvian mining regulations. Pit ramps are 10%. Ramp width is narrower, allowing for single lane traffic, for the final 3 to 4 benches in each pit. Figure 15-3 shows the haul road dimensions for two-way traffic.

Figure 15-3: Haul Road Dimensions

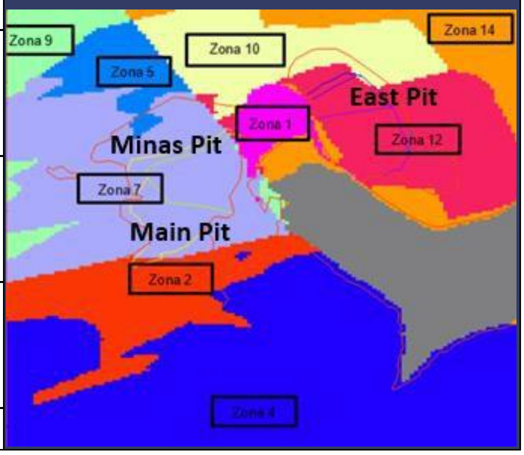
Description	Dimension	CAT 785D
Tyre Model		37.00 R51
Tyre Height (m)	A	3.06
truck Width (m)	B	7.05
Windrow Height (m)	C= 3/4*A	2.30
Windrow Angle		37
Windrow Width (m)	E=C/tan(D)*2	6.09
Transitable Width (m)	F=B*3.0	21.15
Drains (m)	G	0.60
Ramp Width - Theoretical (m)	H=F+E+2*G	28.44
Ramp Width - Design (m)	I=round(H)	29.00



Wall angles used in the design are from the McDonald 2012 and Anddes 2017 geotechnical studies and are shown in Figure 15-4.

Figure 15-4: Wall Angles

Pit - Lithology	Zone	IRA Degrees	Face Angle Degrees	Bench Height m	Berm Width m
All - Bofedal	1	15	37	8.0	19.2
Main - Post Mineralization	15	45	70	16.0	10.2
Main - Pre Mineralisation	2	45	70	8.0	5.1
Main - Other	4	41	63	8.0	5.1
Minas - Post Mineralisation	5	38	58	16.0	10.3
Minas - Pre Mineralisation	7	45	70	8.0	5.2
Minas - Other	9	42	65	8.0	5.2
East - Post Mineralisation	10	46	70	16.0	9.6
East - Pre Mineralisation	12	46	70	8.0	4.8
East – Other	14	46	70	8.0	4.8
Other		46	70	8.0	4.8



Note: IRA = Inter Ramp Angle.

The final pit design contains 1% less ore, 8% more waste, and 4% more total tonnes when compared to the Whittle shell used, as can be seen in Table 15-9.

Table 15-9: Final Pit Design Quantities

Description	Ore Mt	Waste Mt	Total Mt
Whittle Shell 73 RF1.00	145.9	186.92	332.8
Final Pit Design	143.9	02.3	346.1
Design / Whittle	99%	108%	104%

Figure 15-5 shows the final pit design.

Figure 15-5: Final Pit Design Plan

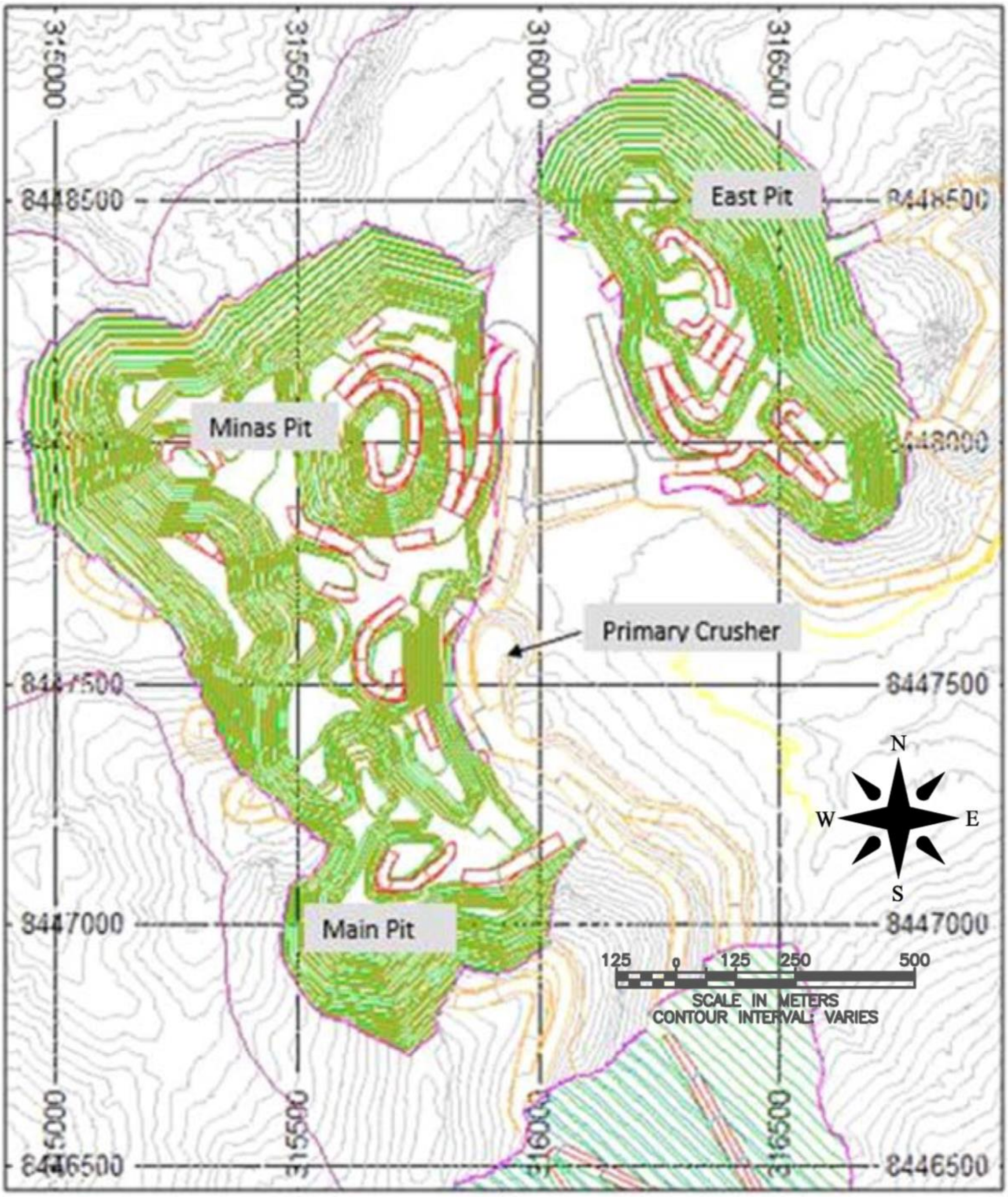


Table 15-10 shows the main dimensions of the final pit.

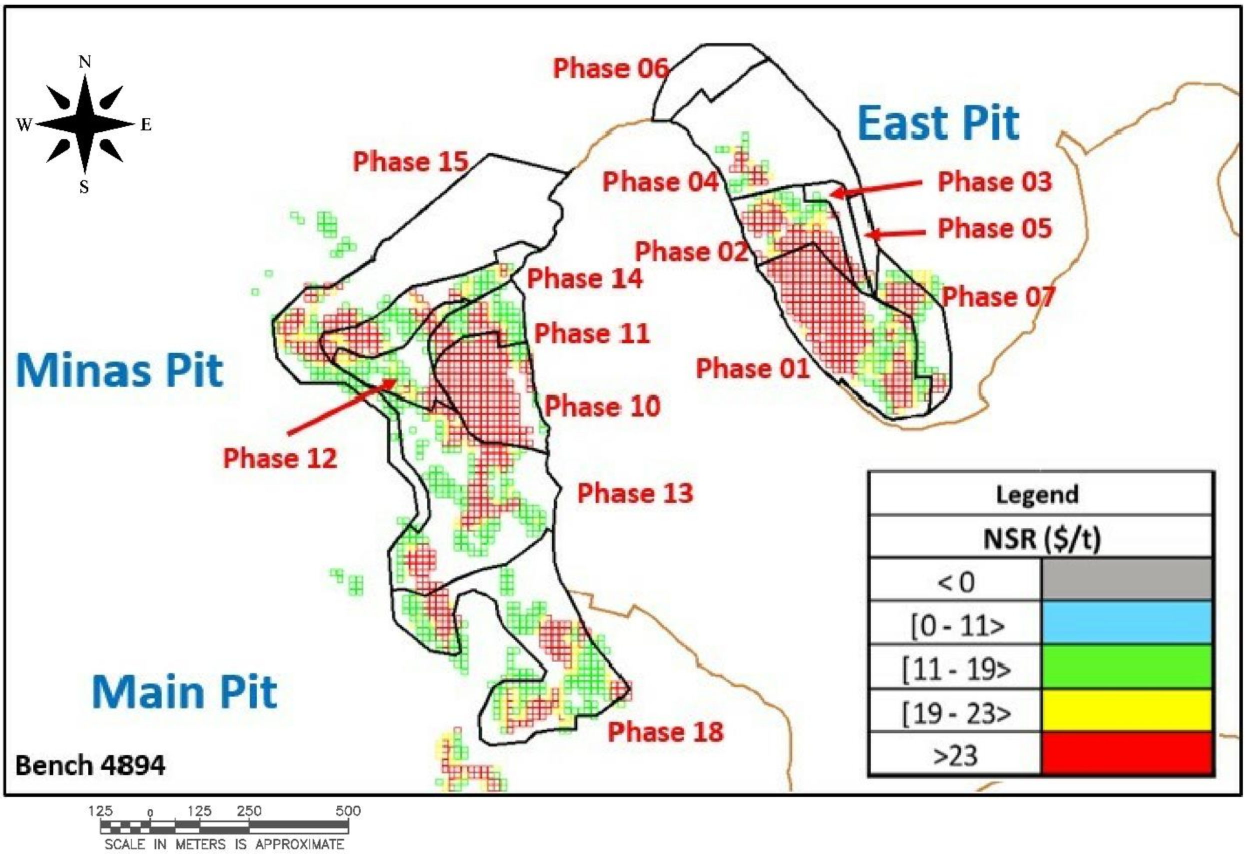
Table 15-10: Dimensions of Final Pit

Description	Unit	Pit		
		East	Minas + Main	Total
Length (max)	m	1,050	1,720	n/a
Width (max)	m	550	1,000	n/a
Area	ha	45.2	105.1	150.3
Highest Point (Crest)	masl	5,050	5,122	5,122
Lowest Point	masl	4,746	4,738	4,738
Height of Highest Wall	m	304	256	304

15.3.2 Internal Phase Designs

To minimize initial pre-stripping requirements and target higher value ore in the early years, a series of internal pit phases has been developed. In total, there are 19 phases (9 at East pit, 7 at Minas pit, 3 at Main pit). The different phases are shown in Figure 15-6, along with the distribution of NSR ($/t) for the bench 4,894 (note some phases are not mines on this particular bench).

Figure 15-6: Pit Phases



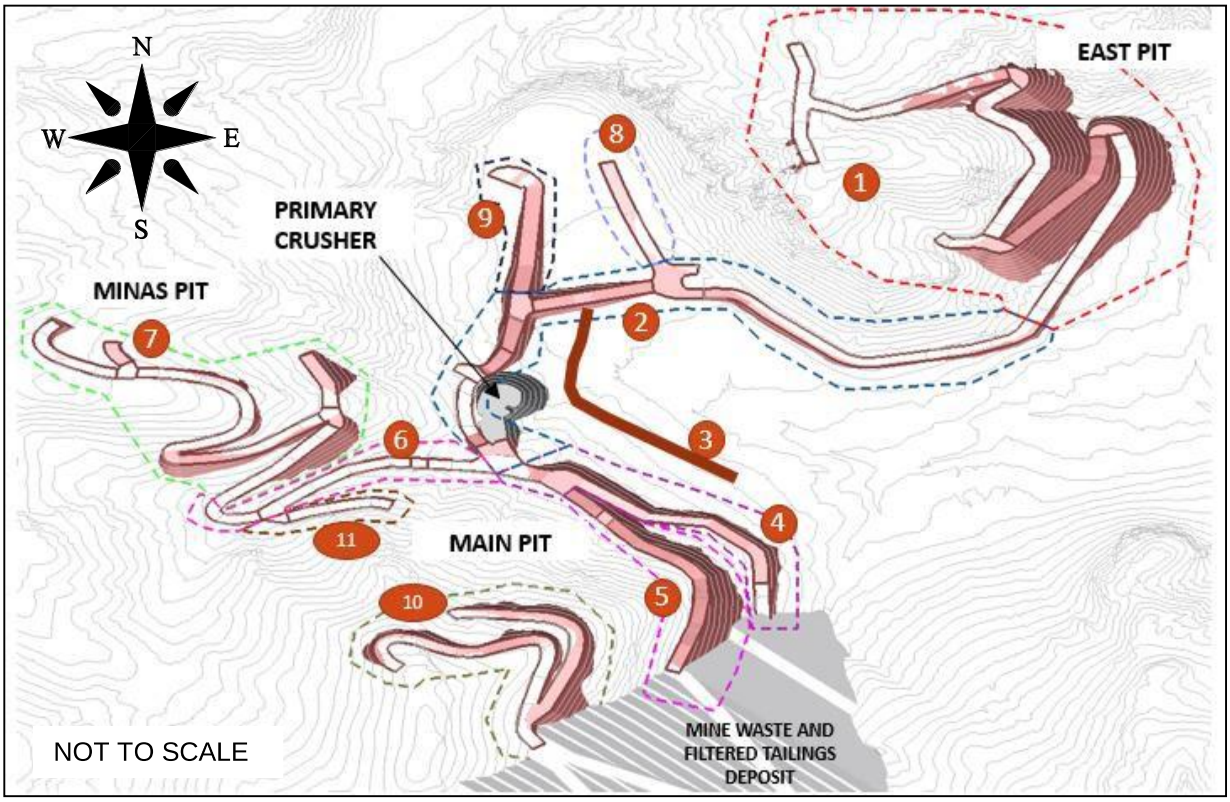
15.3.3 Haul Road Design

A system of haul roads is required to link the open pits to the primary crusher, short term ore stockpile, and the Main Mine Waste and Filtered Tailings Deposit. Haul roads 1,2,3,4,5,6, and 7 (a total of 9.09 km) must be constructed during the pre-strip period. This will involve 1.334 Mt of cut and 5.440 Mt of fill. In addition, the primary crusher pad will require 0.352 Mt of fill during the pre-strip period. The excess fill (i.e., the portion not sourced directly from cut) of 4.558 Mt will be sourced from non-acid generating (NAG) waste from the East pit during the pre-strip period. Haul roads will be finished with 1 m of base where required (estimated at 25% of haul road length, as most haul roads are constructed from mine fill, which will not require a layer of base), and 0.4 m of wearing course. In areas of bofedal, these will be excavated to a depth of up to 3 m and backfilled with layers of mine waste rock wrapped in geo-mesh and geo-textile. Culverts to direct the water from one side of the haul road to the other will be installed at approximately 200-m spacings depending gradient and catchment area.

Four further haul-roads (8, 9, 10, and 11, totaling 2.26 km) are required to be constructed during the operational life of the mine, in years three, five, and 11.

The haul roads are shown on Figure 15-7.

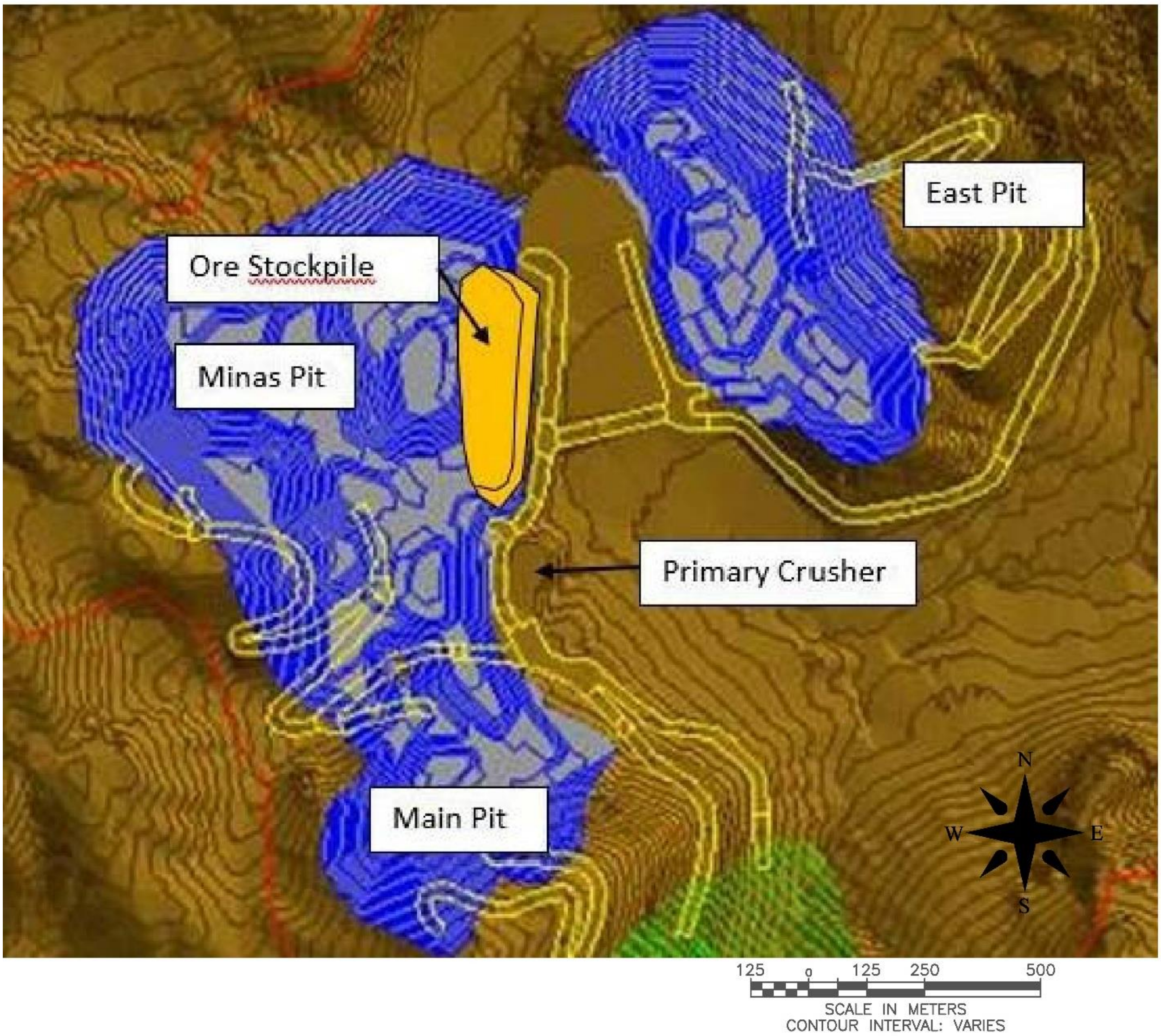
Figure 15-7: Haul Roads



15.3.4 Short Term Stockpile Design

A short-term stockpile will be used during the pre-strip stage and first four years of production. The stockpile is located within the Minas pit limit (as shown in Figure 15-8), and is primarily used to maintain the Zn/Pb ratio above 0.4. The maximum capacity is 2.2 Mt.

Figure 15-8: Short-Term Stockpile Location



15.4 Mineral Reserves

The project Mineral Reserves consider only measured and indicated resource categories, which have been converted to proven and probable reserves categories, respectively. Mineral Reserves are defined as being the material to be fed to the process plant in the mine plan already described, and are demonstrated to be economically viable in the Detailed Design Phase 1 (FEED) economic model. The Mineral Reserves are shown in Table 15-11.

At this time, there are no unique situations relative to mining, metallurgy, permitting, infrastructure, and other relevant factors that would put the Corani mineral reserve at a higher level of risk than any other Peruvian developing resource.

Table 15-11: Corani Project Mineral Reserves

Classification	Tonnes Mt (dry)	Grade			NSR $/t	Contained Metal		
		Silver g/t	Lead %	Zinc %		Silver Moz	Lead Mlb	Zinc Mlb
Proven	20.8	65.8	1.03	0.71	37.17	44	472	323
Probable	118.3	47.5	0.87	0.57	28.55	181	2,274	1,486
Total Proven + Probable	139.1	50.3	0.90	0.59	29.84	225	2,746	1,809

Notes:

1) The Mineral Reserves have been estimated using the definitions of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).
2) The Mineral Reserves have been estimated using the following metal prices: $20.00/oz Ag, $1.00/lb Zn, $0.95/lb Pb using a revenue factor 1.00 pit shell as a basis for the pit design.
3) Only pre-mineral tuff type of material has been considered as reserves.
4) NSR Cut-off grades used are equal or higher than: $11.11/t for the East Pit, and $11.26/t for Minas and Main pits.
5) The effective date for these Mineral Reserves is 1 May 2017.
6) Totals / Averages may not add up due to rounding of individual tonnes and grades.
7) The tonnes and grades shown above are considered a Mineral Reserve because they have been demonstrated to be economically viable through the FEED study financial model using the following metal prices: $18.00/oz Ag, $1.10/lb Zn, $0.95/lb Pb.

The Mineral Reserves from this FEED study differ only very slightly when compared to the Mineral Reserves detailed in the NI 43-101 report (M3 Engineering, 2015): the FEED study reserve tonnes are 101% of the NI43-101 tonnage, whilst the contained silver and lead metal is approximately 1% less, and the contained zinc metal is approximately 1.4% higher. The Mineral Reserves from the NI 43-101 report are shown in the table below.

Table 15-12: 2015 NI43-101 Mineral Reserves

Classification	Tonnes Mt (dry)	Grade			Contained Metal		
		Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Proven	19.855	69.1	1.09	0.72	44.1	478.7	313.4
Probable	117.843	48.6	0.88	0.57	184.3	2,289.2	1,470.7
Proven + Probable	137.698	51.6	0.91	0.59	228	2,768	1,784

Mineral Reserves: from the M3 Engineering NI 43-101 report (2015)

16 Mining Methods

The Corani deposit is planned to be mined using conventional open pit mining methods. The mine design and planning are based on the resource model and reserve estimate as indicated in the previous sections.

16.1 Summary

The mine plan considers extraction of the proven and probable ore material included in the mineral reserve presented in Section 15. The mine plan has been developed to deliver 7,875 ktonnes of ore per year (22,500 tpd x 350 processing days) to the crusher for processing by flotation to produce two concentrates: 1) lead-silver, and 2) zinc-silver.

The mine plan development included the following:

- Ultimate pit design including benches, ramps, and haul roads (Section 15)
- Pit phase design based on the incremental Whittle Shells (Section 15)
- Detailed pit phase designs with benches, ramps, and haul roads (Section 15)
- Phase Design
- Mine production scheduling
- Waste storage design and material allocation
- Time sequence mine plan drawing development
- Equipment and manpower requirement calculations

The following sections detail the development of the mine plan.

16.2 Phase Design and Pre-stripping

Pre-striping of 12.317 Mt, over a period of 14 months, is required to allow sustainable mine production to commence in May 2021, which coincides with the date the process plant will be ready to start its ramp-up. Pre-stripping will be conducted in two distinct phases, with a considerable period between them:

- Phase 1, with construction equipment (10 months, 4.735 Mt). September 2018 to June 2019. The primary purpose is to utilize mine waste to construct access to the process plant site within the project area.
- Phase 2, with the same mining equipment to be employed during the Production period (4 months, 7.582 Mt). January 2021 to April 2021.

16.3 Pre-strip Phase 1, Initial Haul Road and Access Road Construction

Excavation of 4.735 Mt (2.003 Mbcm) of material from the East pit, and a further cut-to-fill volume of 0.605 Mbcm in the haul roads, is required for the construction of haul roads 1 to 7 and filling of the primary crusher pad. This work will be carried out over a 10-month period and includes:

- Topsoil stripping at East Pit Phase 1, haulroads and ore stockpile (14,761 m^3)
- Removal of bofedal for haul road construction (107,136 m^3)
- Placement of large rock fill and installation of geotextile and geo-mesh in areas where haulroads cross bofedals

- Construction of haul road bulk fills using mine waste and cut-to-fill (9.09 km); includes widening of haulroads 1 and 2 to include the mine access road
- Crushing and screening and placement of haul road base (1.0 m) in selected areas, and wearing course (0.4 m) on haul roads
- Lining of an estimated 50% of haul road water tables with high density polyethylene (HDPE) and installation of culverts
- Provision and installation of reflective guide posts

This work is considered as a separate contract, and the resources identified in Table 16-1 will be required.

Table 16-1: Pre-Strip Phase 1 and Construction of Haul Roads: Equipment

Equipment	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18	Jan-19	Feb-19	Mar-19	Apr-19	May-19	Jun-19	Jul-19
Bulldozer CAT D8		4	4	5	4	3	3	4	3	0	0	
Bulldozer CAT D6		0	0	0	0	1	1	1	1	0	0	
Drill ROC L8		2	2	2	1	1	1	1	1	0	0	
Excavator CAT 336		7	7	7	5	5	5	5	5	0	0	
Front-end Loader CAT 966		0	0	0	1	3	2	2	2	3	1	
Grader CAT 140		0	0	0	1	2	3	3	2	2	1	
Tip-truck 8x4		22	22	22	25	32	31	31	15	2	0	
Water Truck 4000 gal		5	5	5	4	4	3	3	3	1	0	
Compactor 19t		1	1	1	1	1	1	1	1	0	0	
Compactor 10t		0	0	0	1	1	2	2	2	2	1	
Primary Crusher		1	1	1	1	1	1	1	1	1	0	
Generator 550 kw		1	1	1	1	1	1	1	1	1	0	
Secondary Crusher		0	0	0	1	1	1	1	1	1	0	
Generator 350 kw		0	0	0	1	1	1	1	1	1	0	
Backhoe CAT 420		0	0	0	4	4	1	1	4	1	1	
Concrete Mixer 9´		0	0	0	0	1	1	1	1	1	0	
Plate Compactor		0	0	0	0	2	2	2	2	2	2	
Total	0	43	43	44	51	64	60	61	46	18	6	0

Personnel requirements range between 35 and 309 during the construction period.

16.4 Owner Operator and Mining Contractor Cases

Two cases have been evaluated for the FEED. Case 1 is where the mining company operates the mine, and Case 2 (the Base Case) is where the mine would be operated by a mining contractor. For the FEED study, it is assumed that general mine coordination, mine planning (including geotechnical and survey), and geology functions would be carried out by the mining company in both cases, which is typical practice in the industry; however, various other responsibility structures are also in use.

16.5 Mine Production and Process Plant Feed Schedules

16.5.1 Mine Production Schedule

Based on the mine design, a mine production schedule which is focused on maximizing project NPV has been developed, following definition of a cut-off grade and stockpiling strategy which was completed (using a slightly different version of the pit design) as part of the FEED study. The three different strategies evaluated were completed as detailed operative mine production schedules using MinePlan software. The results of the three strategies evaluated were compared to the results predicted by Comet software (which was run only for strategy 2 to determine a baseline project value). The three strategies evaluated were:

- Fixed cut-off grade (NSR $/t), with use of short-term ore stockpiles to maintain the Zn/Pb ratio above 0.4.
- Variable cut-off grade (NSR $/t), with use of short-term ore stockpiles to maintain the Zn/Pb ratio above 0.4.
- Variable cut-off grade (NSR $/t), with use of short-term ore stockpiles to maintain the Zn/Pb ratio above 0.4, and long-term stockpiles for lower value material, which is processed at the end of the mine life.

Comet predicted an NPV of $1,418 M (operating cash flow) for strategy 2. The results of the detailed mine scheduling were $1,370 M for strategy 1, $1,424 M for strategy 2, and $1,432 M for strategy 3.

The value of using a variable cut-off grade (NSR $/t) is clearly demonstrated with the 4% increase in operating NPV when a variable cut-off grade is introduced with strategy 2. The use of long-term stockpiles and processing lower value ore at the end of the mine life (strategy 3) adds only another $8M to the VAN when compared to strategy 2. As the risk of reduced metallurgical recovery for long-term stockpiled ore exists, strategy 2 was selected for the final mine production schedule for the FEED study.

The mine schedule includes a pre-strip phase in the East pit, in which 12.311 Mt of waste will be mined. In addition, 20,629 t of ore will be mined and stockpiled during the same period. The total pre-strip requirement is 12.317 Mt. The pre-strip period is broken into a 10-month Phase 1 period and a 4-month Phase 2 period. During the Phase 1 period (September 2018 to June 2019), construction equipment will be used to mine 4.735 Mt of waste which is principally required for haul road and access road construction to gain access to the plant site so that plant construction may commence. In the Phase 2 period (January to April 2021), mining equipment will be used to mine the remaining 7.582 Mt. There is a period of 18 months between the two pre-strip periods when no mining activity is programmed.

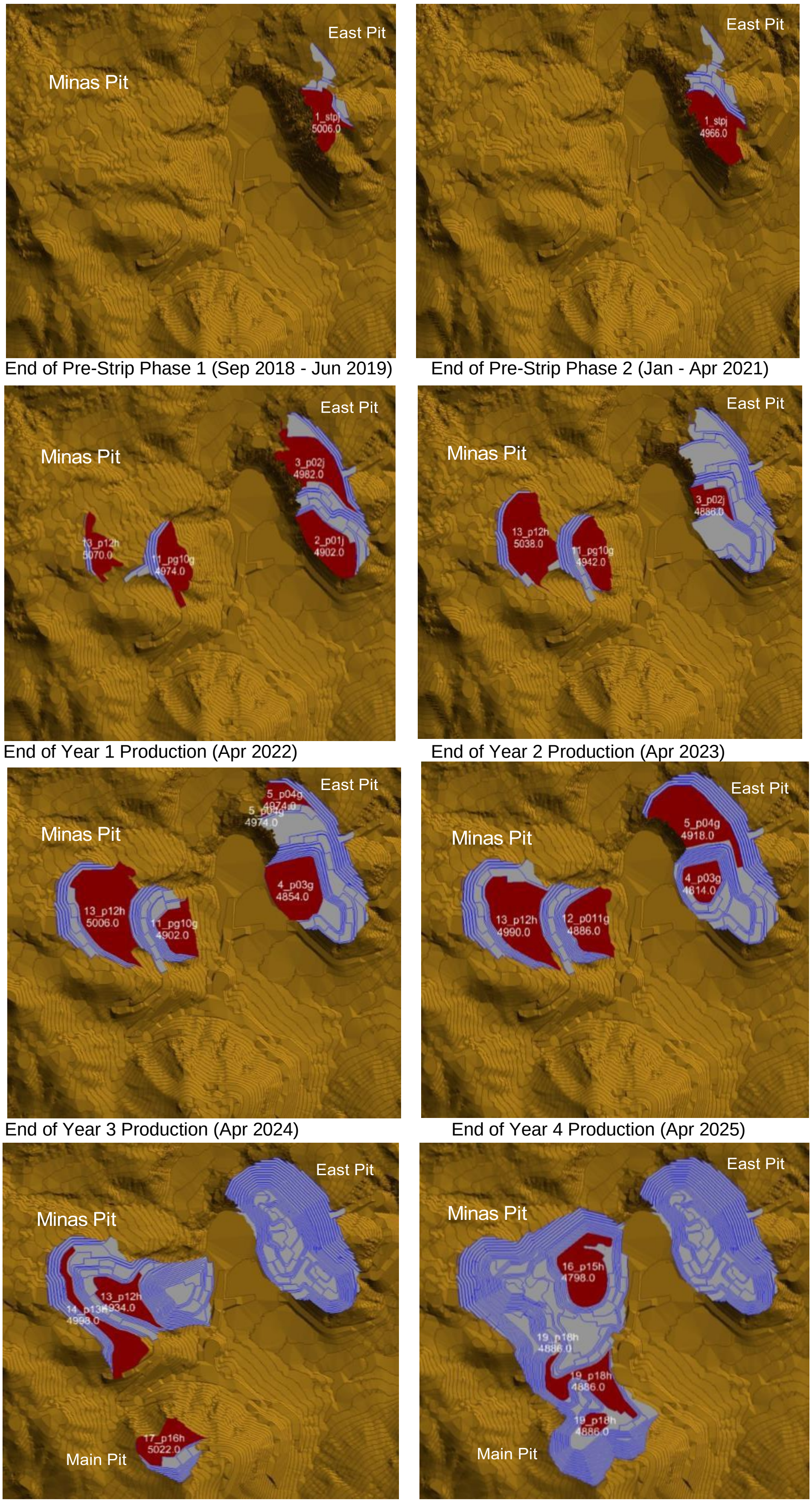
Following completion of the phase 2 pre-stripping, the East pit will be mined continuously until near the end of 2028. Backfilling of the pit with mine waste and filtered tailings will commence in April 2029 and take 4 years to complete.

Mining of Minas pit will commence of November 2021 and will continue continuously until the end of the mine life in April 2039.

Mining at the Main pit commences in 2026 and continues until the end of the mine life in April 2039, except from May 2027 to April 2029 when mining in this pit is not required.

The mine development sequence is shown in the following diagrams.

Figure 16-1: Mine Development Sequence



End of Pre-Strip Phase 1 (Sep 2018 - Jun 2019)
End of Pre-Strip Phase 2 (Jan - Apr 2021)
End of Year 1 Production (Apr 2022)
End of Year 2 Production (Apr 2023)
End of Year 3 Production (Apr 2024)
End of Year 4 Production (Apr 2025)
End of Year 10 Production (Apr 2031)
End of Year 18 Production (Apr 2039)

The mine production schedule is shown in Table 16-2 and Table 16-3.

Table 16-2: Mine Production Schedule (Mt)

Pit	Phase /Period (Mt)	Pre-Strip P1	Pre-Strip P2	May-21 – Apr-22	May-22 – Apr-23	May-23 – Apr-24	May-24 – Apr-25	May-25 – Apr-26	May-26 – Apr-27	May-27 – Apr-28	May-28 – Apr-29	May-29 – Apr-30	May-30 – Apr-31	May-31 – Apr-32	May-32 – Apr-33	May-33 – Apr-34	May-34 – Apr-35	May-35 – Apr-36	May-36 – Apr-37	May-37 – Apr-38	May-38 – Apr-39	Total (Mt)
East	0	4.7	7.6																			12.8
East	1			12.7	6.2																	18.8
East	2			8.0	12.7	1.5																22.2
East	3					4.3	5.2	0.9														10.4
East	4					0.7	11.3	12.2	0.3													24.5
East	5								3.5													3.5
East	6							1.4	8.5	1.6												11.5
East	7								0.4	6.4	1.8											8.6
East	8										3.9											3.9
Minas	9			3.1																		3.1
Minas	10			3.4	4.2	5.6	1.2															14.3
Minas	11						2.7	5.2	1.2	3.2	1.9											14.3
Minas	12			1.5	7.7	11.3	2.0				5.3	7.7	3.2	1.1								39.8
Minas	13											4.8	8.9	14.0	10.3	2.5	1.6					42.1
Minas	14													2.4	5.4	8.0	2.5					18.3
Minas	15																9.6	12.8	9.6	7.2	4.9	44.0
Main	16							1.8	3.0			2.7	1.9	0.1								9.6
Main	17													4.0	1.6							5.6
Main	18														3.2	7.7	5.4	6.5	2.2	4.7	9.0	38.8
Total		4.7	7.6	29.2	30.8	23.3	22.4	21.4	17.0	11.2	13.0	15.2	14.0	21.6	20.5	18.1	19.2	19.3	11.8	11.9	13.9	346.1

Table 16-3: Mine Schedule

Period	Ore Mined To Plant (kt)	Ore Stockpiled (kt)	Ore Mined Total (kt)	Waste Mined NAG (kt)	Waste Mined PAG (kt)	Waste MIned Bofedal (kt)	Waste Mined Total (kt)	Total Mined (kt)	Strip Ratio	Ore Rehandled to Plant (kt)	Total Moved (kt)	Ore Feed to Plant (kt)
Sep-2018 - Jun-2019	0	0	0	4,735	0	0	4,736	4,735	0.00	0	4,735	0
Jan-2021	0	0	0	1,596	0	0	1,596	1,596	0.00	0	1,596	0
Feb-2021	0	0	0	1,988	7	0	1,995	1,995	0.00	0	1,995	0
Mar-2021	0	0	0	1,988	7	0	1,995	1,995	0.00	0	1,995	0
Apr-2021	0	6	6	1,982	7	0	1,989	1,995	312.09	0	1,995	0
May-2021	100	79	179	1,798	15	0	1,813	1,992	10.14	0	1,992	100
Jun-2021	150	274	424	1,533	37	0	1,571	1,995	3.70	0	1,995	150
Jul-2021	185	394	579	1,333	82	0	1,416	1,995	2.44	0	1,995	185
Aug-2021	290	680	970	1,505	185	0	1,690	2,660	1.74	0	2,660	290
Sep-2021	400	0	400	2,192	55	0	2,247	2,647	5.62	0	2,647	400
Oct-2021	355	0	355	2,110	49	0	2,159	2,514	6.08	135	2,649	490
Nov-2021	389	0	389	2,055	56	0	2,111	2,501	5.42	156	2,656	545
Dec-2021	435	0	435	1,995	70	0	2,065	2,499	4.75	150	2,650	585
Jan-2022	411	0	411	1,972	65	0	2,036	2,447	4.95	204	2,651	615
Feb-2022	656	0	656	1,873	116	0	1,989	2,646	3.03	0	2,646	656
Mar-2022	656	0	656	1,858	134	0	1,992	2,648	3.04	0	2,648	656
Apr-2022	656	0	656	1,874	123	0	1,997	2,653	3.04	0	2,653	656

Period	Ore Mined To Plant (kt)	Ore Stockpiled (kt)	Ore Mined Total (kt)	Waste Mined NAG (kt)	Waste Mined PAG (kt)	Waste MIned Bofedal (kt)	Waste Mined Total (kt)	Total Mined (kt)	Strip Ratio	Ore Rehandled to Plant (kt)	Total Moved (kt)	Ore Feed to Plant (kt)
May-2022	656	0	656	1,885	116	0	2,001	2,657	3.05	0	2,657	656
Jun-2022	656	0	656	1,883	117	0	2,000	2,656	3.05	0	2,656	656
Jul-2022	501	0	501	1,900	99	0	1,999	2,501	3.99	155	2,656	656
Aug-2022	515	0	515	1,863	136	0	1,999	2,514	3.88	141	2,655	656
Sep-2022	530	0	530	1,925	72	0	1,997	2,527	3.77	126	2,654	656
Oct-2022	656	0	656	1,902	92	0	1,994	2,650	3.04	0	2,650	656
Nov-2022	656	17	673	1,782	206	0	1,987	2,660	2.95	0	2,660	656
Dec-2022	656	30	686	1,838	136	0	1,974	2,660	2.88	0	2,660	656
Jan-2023	656	45	702	1,676	160	0	1,837	2,538	2.62	0	2,538	656
Feb-2023	656	45	702	1,663	183	0	1,846	2,548	2.63	0	2,548	656
Mar-2023	583	0	583	1,768	114	0	1,882	2,465	3.23	73	2,538	656
Apr-2023	490	0	490	1,819	82	0	1,901	2,391	3.88	166	2,558	656
May-2023	656	0	656	1,280	64	0	1,345	2,001	2.05	0	2,001	656
Jun-2023	656	0	656	1,210	133	0	1,343	2,000	2.05	0	2,000	656
Jul-2023	656	0	656	1,074	260	0	1,334	1,991	2.03	0	1,991	656
Aug-2023	656	0	656	1,044	223	0	1,267	1,923	1.93	0	1,923	656
Sep-2023	656	0	656	1,176	91	0	1,267	1,923	1.93	0	1,923	656
Oct-2023	656	0	656	1,152	118	0	1,270	1,926	1.94	0	1,926	656
Nov-2023	656	0	656	1,049	207	2	1,258	1,914	1.92	0	1,914	656
Dec-2023	656	0	656	1,034	242	3	1,279	1,935	1.95	0	1,935	656
Jan-2024	656	0	656	1,027	239	2	1,268	1,925	1.93	0	1,925	656
Feb-2024	656	0	656	1,056	219	2	1,277	1,934	1.95	0	1,934	656
Mar-2024	656	12	669	1,055	203	2	1,260	1,929	1.88	0	1,929	656
Apr-2024	656	53	710	1,000	218	0	1,219	1,929	1.72	0	1,929	656
May-2024	656	121	777	963	188	0	1,151	1,929	1.48	0	1,929	656
Jun-2024	656	24	680	1,049	200	0	1,248	1,929	1.83	0	1,929	656
Jul-2024	530	0	530	933	111	0	1,044	1,575	1.97	126	1,701	656
Aug-2024	529	0	529	1,161	92	0	1,253	1,782	2.37	127	1,909	656
Sep-2024	543	0	543	1,198	55	0	1,253	1,796	2.31	113	1,909	656
Oct-2024	546	0	546	1,175	84	0	1,259	1,805	2.31	110	1,915	656
Nov-2024	656	0	656	1,138	129	0	1,267	1,923	1.93	0	1,923	656
Dec-2024	656	0	656	1,182	92	0	1,273	1,930	1.94	0	1,930	656
Jan-2025	656	0	656	1,158	111	0	1,269	1,925	1.93	0	1,925	656
Feb-2025	656	0	656	1,158	112	0	1,270	1,926	1.94	0	1,926	656
Mar-2025	656	0	656	1,115	154	0	1,269	1,926	1.93	0	1,926	656
Apr-2025	656	0	656	1,084	194	0	1,279	1,935	1.95	0	1,935	656
May-2025 - Jul-2025	1,969	0	1,969	3,355	779	0	4,134	6,103	2.10	0	6,103	1,969
Ago-2025 - Oct-2025	1,969	0	1,969	3,307	747	0	4,054	6,023	2.06	0	6,023	1,969
Nov-2025 - Ene-2026	1,969	0	1,969	2,458	472	46	2,976	4,945	1.51	0	4,945	1,969
Feb-2026 - Apr-2026	1,969	0	1,969	2,049	326	5	2,380	4,349	1.21	0	4,349	1,969
May-2026 - Jul-2026	1,969	0	1,969	3,303	307	2	3,612	5,581	1.83	0	5,581	1,969
Ago-2026 - Oct-2026	1,969	0	1,969	1,816	313	4	2,133	4,102	1.08	0	4,102	1,969
Nov-2026 - Ene-2027	1,969	0	1,969	1,479	556	9	2,044	4,013	1.04	0	4,013	1,969
Feb-2027 - Apr-2027	1,969	0	1,969	1,095	234	3	1,333	3,301	0.68	0	3,301	1,969
May-2027 - Apr-2028	7,875	0	7,875	778	2,562	2	3,342	11,217	0.42	0	11,217	7,875
May-2028 - Apr-2029	7,875	0	7,875	3,605	1,477	0	5,082	12,957	0.65	0	12,957	7,875
May-2029 - Apr-2030	7,875	0	7,875	7,197	98	0	7,296	15,171	0.93	0	15,171	7,875

Period	Ore Mined To Plant (kt)	Ore Stockpiled (kt)	Ore Mined Total (kt)	Waste Mined NAG (kt)	Waste Mined PAG (kt)	Waste MIned Bofedal (kt)	Waste Mined Total (kt)	Total Mined (kt)	Strip Ratio	Ore Rehandled to Plant (kt)	Total Moved (kt)	Ore Feed to Plant (kt)
May-2030 - Apr-2031	7,875	0	7,875	5,418	748	0	6,166	14,041	0.78	0	14,041	7,875
May-2031 - Apr-2032	7,875	0	7,875	8,686	5,076	0	13,763	21,638	1.75	0	21,638	7,875
May-2032 - Apr-2033	7,875	0	7,875	7,563	5,050	0	12,613	20,488	1.60	0	20,488	7,875
May-2033 - Apr-2034	7,875	0	7,875	7,194	3,059	0	10,253	18,128	1.30	0	18,128	7,875
May-2034 - Apr-2035	7,875	0	7,875	8,355	2,922	35	11,312	19,187	1.44	0	19,187	7,875
May-2035 - Apr-2036	7,875	0	7,875	7,746	3,692	0	11,437	19,312	1.45	0	19,312	7,875
May-2036 - Apr-2037	7,875	0	7,875	1,520	2,419	4	3,943	11,818	0.50	0	11,818	7,875
May-2037 - Apr-2038	7,875	0	7,875	412	3,620	0	4,032	11,907	0.51	0	11,907	7,875
May-2038 - Apr-2039	7,744	0	7,744	624	5,499	0	6,123	13,867	0.79	0	13,867	7,744
Total	137,289	1,783	139,073	160,723	46,221	122	207,064	346,139	1.49	1,783	347,922	139,073

Mining requirements are highest from August 2021 until February 2023 (average 2.586 Mt mined per month), and then reduce over the next 4.5 years to a low of 11.217 Mt in the 12-month period from May 2027 to Apr 2028 (0.935 Mt per month average), then increase again to 21.638 Mt (1.803 Mt per month) in the 12 months from May 2031 to Apr 2032. From 2032 onwards, mining requirements generally reduce to 12 to 14 Mt in the final three years, with mining being completed in April 2039.

16.5.2 Process Plant Feed Schedule

Ore to be fed to the primary crusher of the process plant will be sourced directly from the mine (and also from the short-term stockpile until October 2024). The tonnes and grades to be fed per period are shown in Table 16-4.

Table 16-4: Process Plant Feed Schedule

Period	Ore Sent to Process Plant (kt)	NSR ($/t)	Grade Ag (g/t)	Grade Pb (%)	Grade Zn (%)
May-2021	100	38.49	128.8	1.17	0.04
Jun-2021	150	38.07	115.5	1.15	0.06
Jul-2021	185	39.21	103.6	1.11	0.12
Aug-2021	290	41.76	95.2	1.13	0.30
Sep-2021	400	44.94	91.3	1.12	0.51
Oct-2021	490	44.78	96.4	1.13	0.43
Nov-2021	545	46.70	96.6	1.14	0.53
Dec-2021	585	50.82	97.4	1.18	0.73
Jan-2022	615	49.59	97.9	1.17	0.67
Feb-2022	656	52.51	89.7	1.13	1.02
Mar-2022	656	55.24	90.2	1.12	1.21
Apr-2022	656	54.31	87.5	1.14	1.24
May-2022	656	54.09	82.7	1.15	1.36
Jun-2022	656	58.33	84.0	1.19	1.54
Jul-2022	656	56.03	88.8	1.24	1.30
Aug-2022	656	58.02	90.7	1.27	1.37
Sep-2022	656	52.17	81.2	1.21	1.18
Oct-2022	656	44.99	72.5	1.06	1.05
Nov-2022	656	40.45	78.6	0.96	0.63
Dec-2022	656	39.61	66.7	0.99	0.80

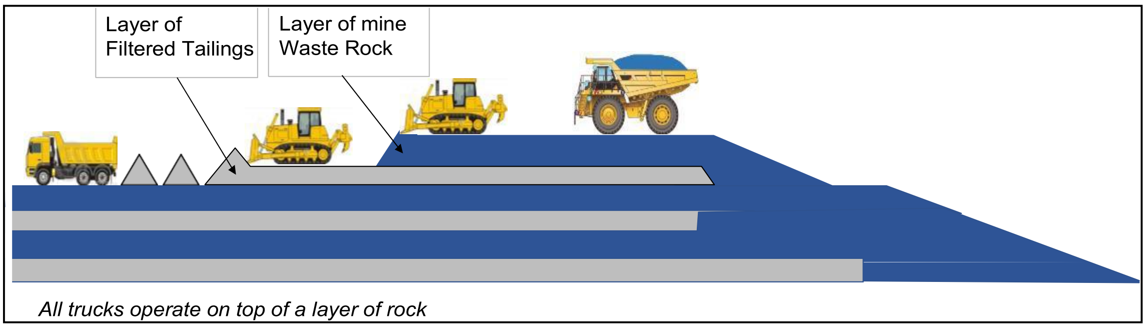
Period	Ore Sent to Process Plant (kt)	NSR ($/t)	Grade Ag (g/t)	Grade Pb (%)	Grade Zn (%)
Jan-2023	656	38.34	63.2	0.93	0.80
Feb-2023	656	35.58	59.4	0.87	0.74
Mar-2023	656	37.66	73.1	0.91	0.57
Apr-2023	656	37.09	83.4	0.95	0.36
May-2023	656	48.32	92.1	1.01	0.59
Jun-2023	656	45.11	88.3	1.09	0.63
Jul-2023	656	53.53	94.9	1.29	0.85
Aug-2023	656	49.91	96.7	1.17	0.72
Sep-2023	656	37.13	80.9	0.85	0.41
Oct-2023	656	42.83	87.0	0.92	0.55
Nov-2023	656	49.38	92.9	1.21	0.76
Dec-2023	656	51.36	92.6	1.26	0.84
Jan-2024	656	58.10	102.9	1.45	0.92
Feb-2024	656	58.06	100.3	1.50	0.93
Mar-2024	656	56.85	95.4	1.56	0.92
Apr-2024	656	55.36	89.9	1.58	0.92
May-2024	656	49.94	81.0	1.52	0.83
Jun-2024	656	54.08	76.7	1.59	0.95
Jul-2024	656	49.03	76.0	1.36	0.84
Aug-2024	656	43.85	71.2	1.02	0.79
Sep-2024	656	42.69	69.7	1.00	0.78
Oct-2024	656	39.62	67.3	0.96	0.70
Nov-2024	656	34.36	63.0	0.91	0.50
Dec-2024	656	32.42	58.1	0.88	0.49
Jan-2025	656	36.68	53.6	1.05	0.63
Feb-2025	656	36.83	53.4	1.06	0.64
Mar-2025	656	36.41	52.3	1.10	0.61
Apr-2025	656	33.67	47.5	1.05	0.56
May-2025 - Jul-2025	1,969	29.36	39.6	1.05	0.49
Aug-2025 - Oct-2025	1,969	26.42	43.3	0.99	0.36
Nov-2025 - Jan-2026	1,969	34.41	65.9	1.04	0.35
Feb-2026 - Apr-2026	1,969	52.88	91.6	1.22	0.86
May-2026 - Jul-2026	1,969	41.95	73.6	1.15	0.71
Aug-2026 - Oct-2026	1,969	44.11	67.6	1.21	0.89
Nov-2026 - Jan-2027	1,969	41.67	63.7	1.10	0.79
Feb-2027 - Apr-2027	1,969	50.22	71.6	1.11	1.24
May-2027 - Apr-2028	7,875	25.02	28.4	0.80	0.73
May-2028 - Apr-2029	7,875	22.74	38.6	0.70	0.58
May-2029 - Apr-2030	7,875	20.84	60.0	0.89	0.16
May-2030 - Apr-2031	7,875	21.96	56.6	0.96	0.23
May-2031 - Apr-2032	7,875	20.87	40.2	0.81	0.32
May-2032 - Apr-2033	7,875	24.68	34.9	0.70	0.74
May-2033 - Apr-2034	7,875	23.76	36.9	0.77	0.59
May-2034 - Apr-2035	7,875	23.00	37.6	0.75	0.50
May-2035 - Apr-2036	7,875	22.41	36.0	0.66	0.49
May-2036 - Apr-2037	7,875	22.51	33.5	0.74	0.45
May-2037 - Apr-2038	7,875	24.26	31.3	0.90	0.52
May-2038 - Apr-2039	7,744	24.63	26.7	0.74	0.79
Total	139,073	29.84	50.3	0.90	0.59

16.6 Waste Management

Mine waste is classified as either non-acid generating (NAG) or potentially acid generating (PAG) based on the Net Neutralizing Potential (NNP) field in the resource block model supplied by Bear Creek, and applying the same criteria as used in the NI 43-101 report (M3 Engineering, 2015), where blocks with NNP values less than -20 are considered to be PAG and blocks with more than -20 are considered to be NAG. In total, 78% of the waste to be mined is classified as NAG. During the pre-strip period, 99.8% of the waste to be mined is NAG, and this will be used to construct the initial haul roads, primary crusher platform, and the platform of the Main Mine Waste and Filtered Tailings Deposit.

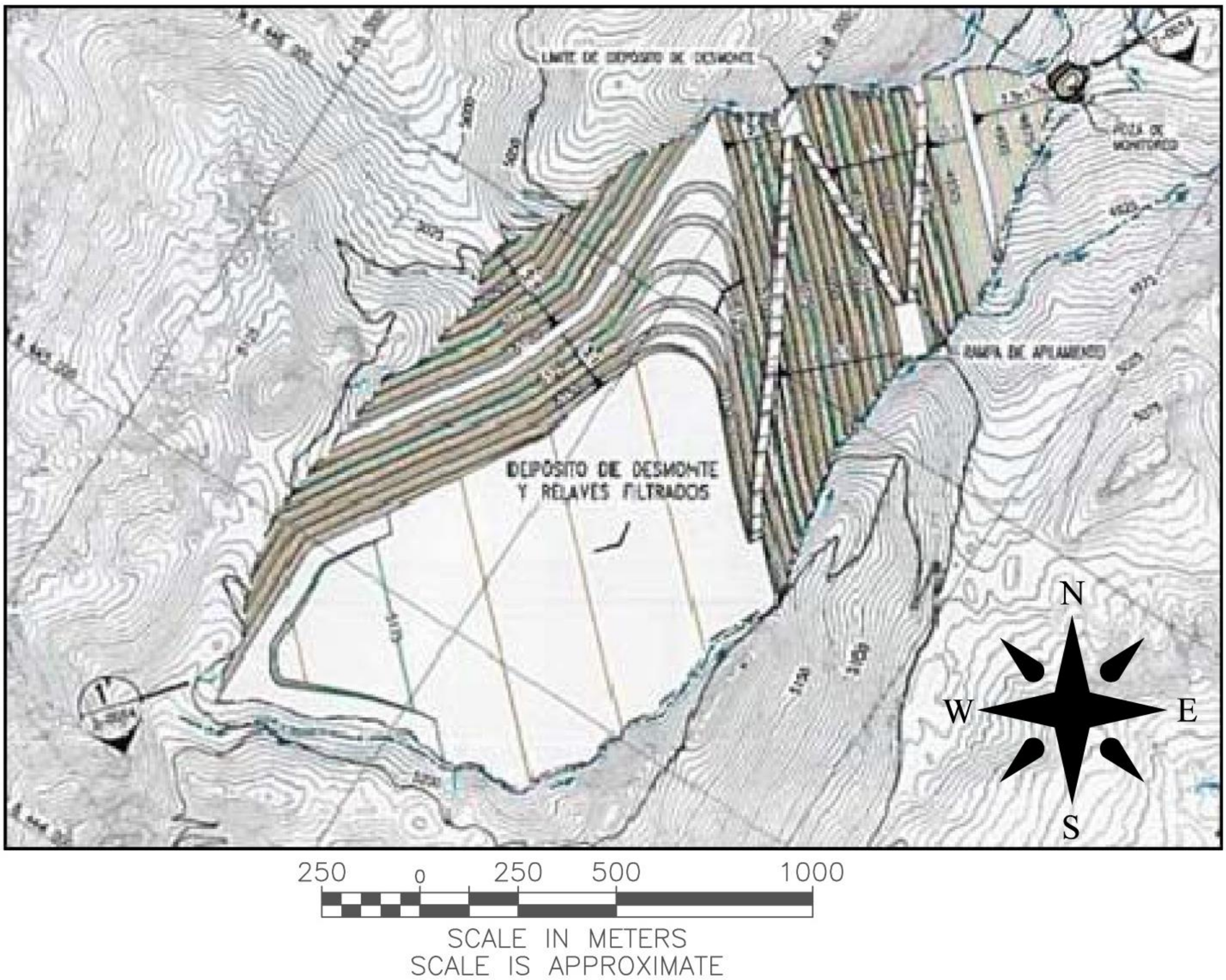
During the production period, waste rock will be used to construct the Main Mine Waste and Filtered Tailings Deposit, using NAG material on the outside faces of the deposit, and using both NAG and PAG to form layers of rock which will cover each layer of filtered tailings, as shown in Figure 16-2.

Figure 16-2: Diagram of Main Mine Waste and Filtered Tailings Deposit Construction



The Main Mine Waste and Filtered Tailings Deposit were designed by Anddes and are discussed in more detail in Section 20 of this report. Generally, the downstream toe will be at approximately 4,856 m.a.s.l. and the top will be at 5,172 m.a.s.l., giving an overall height of approximately 316 m. Above the initial platform, face angles of 2.2 H:1 V will be used, with 7.5-m wide berms spaced every 10 m vertical, giving an overall angle of 3 H:1V. The downstream face will be profiled and rehabilitated progressively during the mine life. The layout is show in Figure 16-3.

Figure 16-3: Main Mine Waste and Filtered Tailings Deposit



Following completion of mining at the East pit in April 2029, backfilling of this pit will take place during the following four years. 32.1 Mt of mine waste (NAG and PAG) and 11.1 Mt of filtered tailings will be used to backfill the pit to 4,958 m.a.s.l. A small quantity of filtered tailings and mine waste will continue to be sent to the Main Mine Waste and Filtered Tailings Deposit during this time, and from 2033 until the end of mining in April 2039 will be the only destination for mine waste and filtered tailings.

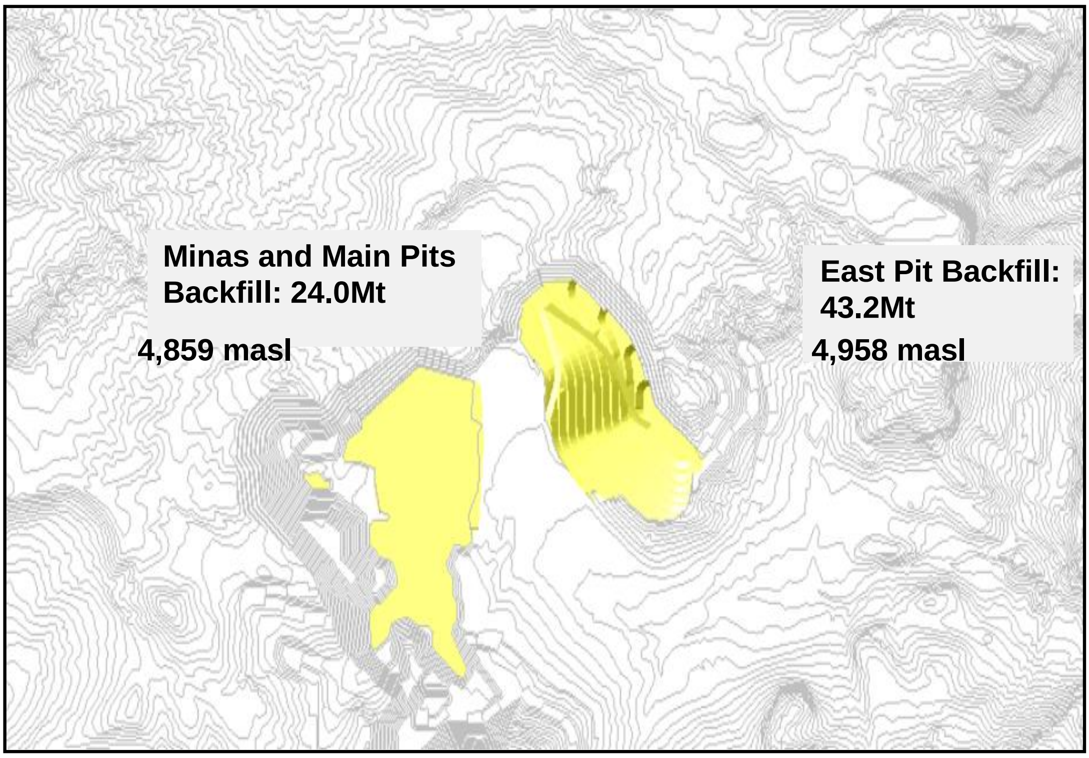
Table 16-5 shows the design criteria for the East pit backfill.

Table 16-5: East Pit Backfill Design Criteria

Parameter	Units	Criterion
Volume	Mm3	25.4
Density (Average)	t/m3	1.70
Capacity	Mt	43.2
Batter Angle	Degrees	24.4
Overall Angle	Degrees	18.4
Overall Angle	H:V	3:1
Lift Height	m	10.0
Berm Width	m	7.5
Ramp Width	m	29.0
Ramp Grade	%	10.0

As part of the mine closure activities, following completion of mining in April 2039, Minas and Main pits will be backfilled to 4,859 masl approximately using 24 Mt of waste (NAG and PAG) and filtered tailings rehandled from the top of the Main Mine Waste and Filtered Tailings Deposit. Planning in the last few years of the mine life will be important to ensure that a layer of NAG waste is located within the Main Mine Waste and Filtered Tailings Deposit so that at completion of backfilling activities, a 1-m minimum layer of NAG waste rock is left on the top of the deposit for closure purposes. Figure 16-4 shows the backfill of the both pits.

Figure 16-4: Pit Backfill



16.7 Mining Equipment

16.7.1 Drilling

Drilling will be carried out using drill rigs equipped with down-the-hole hammers. Based on the rock hardness documented in the report “Corani Blast Fragmentation Project, Topex” (2015), the rock is of low to medium hardness with an unconfined compressive strength ranging between 27.5 MPa and 36.9 Mpa. The drill pattern selected allows for a powder factor of 0.26 kg/t, consistent with the requirement documented by Topex.

Drilling will be conducted using two Atlas Copco DM45 drills, with 156-mm diameter holes, using an average pattern as shown in the following table. These drills will work 169,464 hours during the life of the mine. One replacement will need to be purchased, resulting in the average hours for each drill being 56,488 over the life of the project.

Table 16-6: Drill and Blast Parameters

Parameter	Unit	Ore	Waste
Density (Average)	t(dry)/bcm	2.40	2.31
Burden	m	4.40	4.40
Spacing	m	5.06	5.06
Bench Height	m	8.0	8.0
Subdrill	m	0.8	0.8

Parameter	Unit	Ore	Waste
Total m Drilled per Hole	m	8.8	8.8
Hole Diameter	Inch	6 1/8	6 1/8
Hole Diameter	m	0.156	0.156
Stemming	m	3.11	3.11
Volume per Hole	bcm/Hole	178	178
Buffer	%	4.0%	4.0%
Re-drills	%	4.0%	4.0%
Penetration Rate	m/h(SMU)	45.0	45.0
Meters Drilled per Hole (excl Buffer, Re-drills)	m/Hole	8.80	8.80
Meters Drilled per Hole (incl Buffer, Re-drills)	m/Hole	9.50	9.50
% ANFO	%	50%	50%
% HANFO 60/40	%	50%	50%
Density ANFO	g/cm3	0.80	
Density HANFO 60/40	g/cm3	1.25	
ANFO per Meter Charged	kg/m	7.6	7.6
HANFO 60EM/40AN por Meter Charged	kg/m	11.9	11.9
Explosives per Hole ANFO	kg/Hole	43.3	43.3
Explosives per Hole HANFO 60/40	kg/Hole	67.6	67.6
Explosives per Hole Total	kg/Hole	110.8	110.8
Powder Factor	kg/bcm	0.62	0.62
Powder Factor	kg/t(dry)	0.26	0.27

A third drill will be required during 2021 and the first half of 2022 when production requirements are higher. The third drill selected is a Montabert top-hammer drill mast-mounted on a small excavator, and will drill 114-mm diameter holes. This drill has been selected as it can also be utilized for drilling of wall control holes, secondary basting, and probe drilling to confirm locations and safe cover of underground workings. Hours programmed for this drill are 24,727, and it will need to be replaced once during the mine life.

Presplit drilling is not envisaged; instead, wall control blasting techniques will be developed using buffer or cushion holes in blasts which are against final pit walls (and phase pit walls) where required.

16.7.2 Blasting

Blasting activates will be undertaken by a specialist blasting contractor, who will provide the explosives and a “down-the-hole” service, whereby the contractor is responsible for designing drill patterns (with approval by the client), priming, loading, and firing of each blast. As some holes will be wet (rainfall and ground water), it is estimated that 50% of blasting will utilize ANFO (for dry holes) and 50% will utilize a waterproof blend of emulsion and ANFO. Explosives will be loaded into drill holes using a Mobile Mixing Unit Truck provided by the contractor. Blasting is expected to be conducted on average three times per week. All personnel within a radius of 500 m of the blast must be evacuated during firing time, as required by the Peruvian mining regulations.

Blasting accessories to be used include boosters (1 lb), downhole non-electric detonators, surface delay non-electric detonators, detonating cord, and non-electric lead-in line. To ensure acceptable blasting performance, crushed stemming will be used.

16.7.3 Loading, Hauling, and Ore Rehandling

In addition to the mining requirements, some ore will need to be rehandled to the primary crusher from the short-term stockpile (located within the Minas Pit limit), during 2021 to 2024. In addition to this rehandle, and due to mining requirements never matching exactly the primary crusher requirements, an estimated 15% of the ore mined will require rehanding. This ore will be stockpiled either on the short-term stockpile (until 2024), adjacent to the primary stockpile, or within the final limits of the pits.

Production loading will be undertaken using two Caterpillar 6030 face shovels, loading Caterpillar 785D dump trucks. A Caterpillar 993K front-end loader will also be required for loading of rehandled ore into the dump trucks and to allow loading activities to start up again quickly after blasting.

The characteristics of the loading and hauling equipment are shown in Table 16-7 and Table 16-8.

Table 16-7: Loading Equipment

Parameter	Unit	Equipment Make/Model		
		Face Shovel Caterpillar 6030	Face Shovel Caterpillar 6030	Front-end Loader Caterpillar 993K (Highlift)
		Material		
		Ore 2.40	Waste 2.31	Ore Rehandle
Density (Average) Moisture Content	t(dry)/bcm)	2.40	2.31	-
Moisture Content	%	5	5	5
Loose Density (Average)	t(wet)/bcm)	1.938	1.866	1.938
Operating Weight	t	294	294	134
Net Power	kW	1,140	1,140	775
Bucket Capacity (Heaped 2:1)	m3	15.0	15.0	13.0
Bucket Capacity (Max payload)	t(wet)/pass	30.0	30.0	22.7
Bucket Fill Factor	%	90%	90%	90%
Bucket Capacity (at Bucket Fill Factor)	t(wet)/pass	25.8	25.8	22.3
Number of Passes to Fill Cat 785D Truck	#	5	6	6
Loading Cycle Time	min	2.75	3.25	4.35
Physical Availability	%	90.0%	90.0%	90.0%
Utilisation of Available Time (Max Possible)	%	85.4%	85.4%	85.4%
Loading Time per Day (Max Possible)	h	20.5	20.5	20.5
Productivity - Maximum - LOM	t(wet)/h(SMU)	2,583	1,632	1,662
Productivity - Minimum - LOM	t(wet)/h(SMU)	2,233	2,232	1,662
Productivity - Average - LOM	t(wet)/h(SMU)	2,558	2,106	1,662

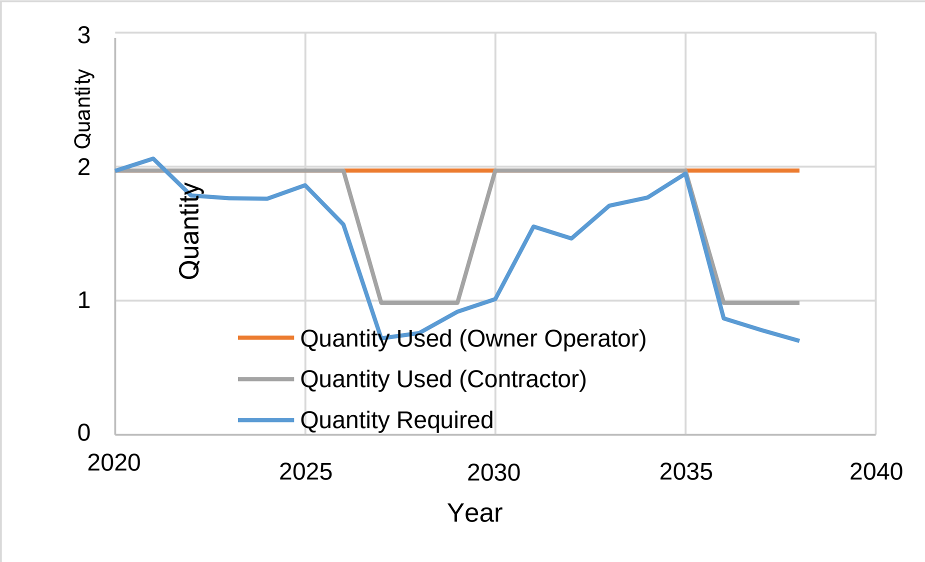
Table 16-8: Hauling Equipment

Parameter	Unit	Equipment Make/Model		
		Dump Truck - Cat785D	Dump Truck - Cat785D	Dump Truck - Cat785D
		Loader - Caterpillar 6030FS	Loader - Caterpillar 6030FS	Loader - Caterpillar 993K
Gross Vehicle Weight	t	249	249	134
Net Power	kW	1,005	1,005	775
Tray Capacity (Nominal)	t(wet)	136.0	136.0	136.0
Payload Average (Matched to Loader Passes)	t(wet)	130.8	136.0	136.0
Physical Availability	%	90.0%	90.0%	90.0%
Utilisation of Available Time (Max Possible)	%	85.4%	85.4%	85.4%
Productivity - Maximum - LOM	t(wet)/h(SMU)	722	417	731
Productivity - Minimum - LOM	t(wet)/h(SMU)	256	184	731
Productivity - Average - LOM	t(wet)/h(SMU)	490	255	731

Shovel requirements vary between a short-term peak of 2.09 in 2021 and 0.73 in 2027. Total shovel hours required are 168,627. Under the Owner Operator scenario, two shovels would need to be purchased for the start of the phase 2 pre-strip. Each shovel will work 84,314 hours during the mine life and will not need replacing. Under the Mining Contractor scenario, two shovels will be employed, except for the 3-year period from 2027 to 2029, where only one would be assigned to the project. The shovel requirements are illustrated in Figure 16-5.

For both scenarios, Owner Operator and Mining Contractor, one caterpillar 993K front-end loader will be required for rehandle of ore. The front-end loader will work 13,963 hours over the life of the mine.

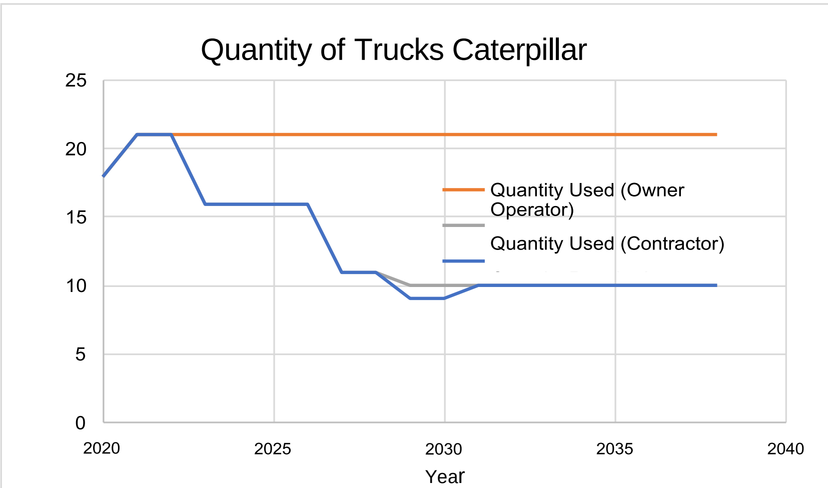
Figure 16-5: Quantity of Shovels – Caterpillar 6030



A total of 1,167,195 dump truck hours are required during the life of the mine (Phase 2 pre- strip and production). Initially, 18 trucks are required for 2020, increasing to 21 for 2021 and 2022, then reducing to 16 in 2023, and reducing further to a minimum of 9 in 2029 and 2030. For 2031 to 2038, 10 trucks are required. The quantity of dump trucks required and used is shown in Figure 16-6.

For the Owner Operator scenario, Bear Creek would have to purchase 18 dump trucks for 2020 and a further 3 (totalling 21) for 2021. These 21 trucks would operate for the rest of the mine life, but with low utilization. Each truck would work 55,580 hours during the mine life and will not need to be replaced.

Figure 16-6: Quantity of Trucks – Caterpillar 785D



For the Mining Contractor scenario, it is assumed that a contractor would have the resources to cover the peak demands and adjust truck numbers to those required.

16.7.4 Ancillary Work and Equipment

Typical ancillary activities will be required to support drilling, loading and hauling activities. These include maintaining floors in the loading and unloading areas, scaling pit walls, construction of temporary ramps and accesses, maintaining haul roads, spreading waste, and profiling the external faces of the Main Mine Waste and Filtered Tailings Deposit, and well as maintenance of short-term ore stockpiles. Ancillary equipment requirements are indicated in Table 16-9.

Table 16-9: Ancillary Equipment Requirements

Equipment	Make	Model	Quantity	Hours Each	Hours Total	Useful Life
Bulldozer	Caterpillar	D10T2	2	50,019	100,038	60,000
Grader	Caterpillar	16M3	1	99,716	99,716	35,000
Wheeldozer	Caterpillar	834K	1	78,400	78,400	35,000
Compactor	Caterpillar	CS54	1	56,458	56,458	18,000
Water Truck	Caterpillar	773G	1	39,075	39,075	50,000
Backhoe	Caterpillar	420F	1	56,000	56,000	18,000
Excavator	Caterpillar	374FL	1	56,000	56,000	12,000
Fuel / Lube Truck	TBD	TBD	2	Contracted Service		

16.7.5 Dewatering

Based on the 2015 Feasibility Study, groundwater inflow to the pit is anticipated to be on the order of 2,000 to 3,000 m^3/day. Inflowing groundwater and water from direct precipitation in the pits will be collected in sumps. Sump locations will change as the pits develop, moving to the deepest portions of the pits as they develop. Water from the pits will be pumped to the contact-water cell of the water supply pond for subsequent consumption during operations.

Pumps will be trailer-mounted, diesel units, and will pump through HDPE pipes to the process plant contact water pond. A range of pump sizes will be required, including some small portable pump and “layflat” hose for transferring water accumulated in surface puddles to a central sump.

16.7.6 Mine Supervision

Mine supervision includes the following areas:

- Mining Operations
- Planning
- Geology
- Maintenance

Table 16-10 identifies the quantity of mine supervision personnel needed and compares the numbers from the 2015 Feasibility Study to the current estimates for the Owner Operator and Mining Contractor scenarios.

Table 16-10: Comparison of Mining Supervision Requirements (Phase 2 Pre-strip and Operations). Feasibility Study 2015 Vs Case 1 Owner Operator Vs Case 2 Mining Contractor.

Position	Area	Quantity (Max)		
		FS 2015	FEED (Owner Operator)	FEED (Mining Contractor)
Mine Manager	Mine Operations	1	1	1
Mine Superintendent	Mine Operations	2	1	1
Mine Foreman	Mine Operations	4	4	0
Foreman	Mine Operations	-	3	0
Engineer	Mine Operations	4	-	1
D&B Engineer	Mine Operations	-	1	0
Heavy Equipment Trainer	Mine Operations	-	1	0
Heavy Equipment Evaluator	Mine Operations	-	1	0
Heavy Equipment Training Assistant	Mine Operations	-	1	0
Chief Mine Engineer	Planning	4	1	1
Geotechnical Engineer	Planning	-	1	1
Surveyor	Planning	2	2	2
Mine Planning Engineer ST	Planning	-	2	2
Mine Planning Engineer LT	Planning	-	1	1
Chief Geologist	Geology	4	1	1
Geologist	Geology	4	-	0
Geologist - Field	Geology	-	2	2
Geologist - Grade Control Modeling	Geology	-	2	2
Senior Geologist - Resources	Geology	-	1	1
Database Administrator	Geology	-	1	1
Database Assistant	Geology	-	2	2
Maintenance Superintendent	Maintenance	2	1	0
Maintenance Chief	Maintenance	-	1	0
Maintenance Foreman	Maintenance	4	-	0
Maintenance Shift Supervisor	Maintenance	-	4	0
Supervisor - Tires	Maintenance	-	1	0
Chief Maintenance Planner	Maintenance	-	1	0
Supervisor - Monitoring	Maintenance	-	3	0
Maintenance Analyst	Maintenance	-	2	0
Maintenance Programmer	Maintenance	-	2	0
Tailings Transport Supervisor	Tailings Transport	-	3	3
Subtotal - Owner		31	47	22
Project Manager	Mining Contractor	-	-	1
Mine Superintendent	Mining Contractor	-	-	1
Shift Supervisor	Mining Contractor	-	-	3
Shift Supervisor - Assistant	Mining Contractor	-	-	3
D&B Engineer	Mining Contractor	-	-	1
Heavy Equipment Trainer	Mining Contractor	-	-	1
Heavy Equipment Evaluator	Mining Contractor	-	-	1
Heavy Equipment Training Assistant	Mining Contractor	-	-	1
Chief - Technical Office	Mining Contractor	-	-	1
Contract Administrator	Mining Contractor	-	-	1
Assistant	Mining Contractor	-	-	2
Engineer - Planning & Costs	Mining Contractor	-	-	1
Surveyor	Mining Contractor	-	-	1
SSOMA: Jefe de SSOMA	Mining Contractor	-	-	1
SSOMA: Ing SSOMA	Mining Contractor	-	-	3
Project Administrator	Mining Contractor	-	-	1

Position	Area	Quantity (Max)		
		FS 2015	FEED (Owner Operator)	FEED (Mining Contractor)
Chief - Logistics and Warehouse	Mining Contractor	-	-	1
Warehouse Assistant	Mining Contractor	-	-	1
Social Assistant	Mining Contractor	-	-	3
Payroll Assistant	Mining Contractor	-	-	2
Chief - Human Resources	Mining Contractor	-	-	1
Community Relations Assistant	Mining Contractor	-	-	1
General Services Coordinator	Mining Contractor	-	-	1
IT Coordinator	Mining Contractor	-	-	1
Driver	Mining Contractor	-	-	3
Maintenance Superintendent	Mining Contractor	-	-	1
Maintenance Chief: Equipment	Mining Contractor	-	-	1
Maintenance Shift Supervisor	Mining Contractor	-	-	4
Supervisor - Tires	Mining Contractor	-	-	1
Maintenance Chief: Planning & Reliabili	Mining Contractor	-	-	1
Maintenance Reliability	Mining Contractor	-	-	3
Maintenance Planner	Mining Contractor	-	-	2
Maintenance Programmer	Mining Contractor	-	-	2
Subtotal - Mining Contractor	All	-	-	52
Total - Owner + Mining Contractor		31	47	74

16.7.7 Principal and Ancillary Equipment

List of Principal and Ancillary Equipment – Construction (Pre-strip Phase 2) and Operations. Case 2 Mining Contractor

Note: All of January 2021 (21m1) to April 2021 (21m4) is Construction. May 2021 (21m5) onwards is Operations.

Table 16-11: List of Principal and Ancillary Equipment - Case 2 Mining Contractor

Equipment Make	Model	21m1	21m2	21m3	21m4	21m5	21m6	21m7	21m8	21m9	21m10	21m11	21m12	22m1	22m2	22m3	22m4	22m5	22m6	22m7	22m8	22m9	22m10	22m11	22m12
Drill - Caterpillar	DM45	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Drill - Montabert	TBD	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shovel - Caterpillar	6030	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Front-end Loader – Caterpillar	993K	-	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Dump Truck - Caterpillar	785D	18	18	18	18	18	20	20	20	20	20	20	20	20	20	21	21	21	21	21	21	20	16	16	16
Bulldozer – Caterpillar	D10T2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Grader - Caterpillar	16M3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel Dozer - Caterpillar	834K	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Compactor - Caterpillar	SC54	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck - Caterpillar	773G	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backhoe - Caterpillar	420F	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel / Lube Truck – TBD	TBD	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Excavator - Caterpillar	374FL	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total		33	33	33	33	34	36	36	36	36	36	36	36	36	36	37	37	37	37	37	37	36	32	32	32

Equipment Make	Model	23m1	23m2	23m3	23m4	23m5	23m6	23m7	23m8	23m9	23m10	23m11	23m12	24m1	24m2	24m3	24m4	24m5	24m6	24m7	24m8	24m9	24m10	24m11	24m12
Drill - Caterpillar	DM45	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Drill - Montabert	TBD	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shovel - Caterpillar	6030	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Front-end Loader – Caterpillar	993K	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Dump Truck - Caterpillar	785D	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16
Bulldozer - Caterpillar	D10T2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	2	2
Grader - Caterpillar	16M3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel Dozer - Caterpillar	834K	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Compactor - Caterpillar	SC54	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck - Caterpillar	773G	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backhoe - Caterpillar	420F	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel / Lube Truck - TBD	TBD	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Excavator - Caterpillar	374FL	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total		32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	31	32	32

Equipment Make	Model	25m1	25m2	25m3	25m4	25m5 – 25m7	25m8 – 25m10	25m11-26m1	26m2-26m4	26m5-26m7	26m8-26m10	26m11-27m1	27m2-27m4	27m5-28m4	28m5-29m4	29m5-30m4	30m5-31m4	31m5-32m4	32m5-33m4	33m5-34m4	34m5-35m4	35m5-36m4	36m5-37m4	37m5-38m4	38m5-39m4
Dril - ICaterpillar	DM45	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	2	2	2	2	2	1	1	1
Drill - Montabert	TBD	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shovel - Caterpillar	6030	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	2	2	2	2	2	2	1	1	1
Front-end Loader – Caterpillar	993K	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Dump Truck - Caterpillar	785D	16	16	16	16	16	16	16	16	16	11	11	11	11	11	10	10	10	10	10	10	10	10	10	10
Bulldozer - Caterpillar	D10T2	2	2	2	2	2	1	1	1	2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader - Caterpillar	16M3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel Dozer - Caterpillar	834K	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Compactor - Caterpillar	SC54	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck - Caterpillar	773G	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backhoe - Caterpillar	420F	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel / Lube Truck – TBD	TBD	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Excavator - Caterpillar	374FL	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total		32	32	32	32	32	31	31	31	32	26	26	24	24	24	23	24	25	25	25	25	25	23	23	23

16.7.8 Mine Operations Personnel

In the scenario that mining is carried out by Bear Creek Mining, a total of 47 staff would be required for the five areas (Mine Operations, Geology, Planning, Maintenance, and Tailings Transport). If a mining contractor were to be used, the quantity of owner staff in these areas would reduce to 22, and the contractor would require 52 staff. Under the mining contractor scenario, planning, geology, and filtered tailings transport supervision would remain with Bear Creek Mining, and the mining contractor would be responsible for mine operations and maintenance, with Bear creek maintaining a reduced staff level in mine operations to coordinate the mining contractor. Because the mining contractor includes staff for functions such as safety supervision, payroll, human resources etc., it is expected that a significant reduction could be made in quantities of Bear Creek Mining staff in its corresponding areas. Table 16-12 shows the Mine Operations personnel requirements and the comparison between the 2015 Feasibility Study and the current Owner Operator and Mining Contractor scenarios.

Table 16-12: Mine Operations Personnel Requirements

Area	Quantity (Max)		
	FS 2015	FEED (Owner Operator)	FEED (Mining Contractor)
Mine Operations	11	13	3
Geology	6	7	7
Planning	8	9	9
Maintenance	6	15	0
Tailing Transport	-	3	3
Subtotal - Owner	31	47	22
Mining Contractor	-	-	52
Total	31	47	74

16.7.9 Filtered Tailings Transport

Transport of filtered tailings from the filtered tailings stockpile to the Main Mine Waste and Filtered Tailings Deposit will be undertaken by a contractor. The contractor will be responsible for construction of its own facilities (with the exception of a wash bay) on a platform to be provided by Bear Creek Mining, and connection to services provided by Bear Creek.

The contractor proposes to use two Caterpillar 966 front-end loaders to load tip-trucks of 20-m^3 capacity, which will transport the filtered tailings to the deposit. The quantity of tip trucks will increase gradually from 6 in the first year to 16 in the last 5 years as haulage cycle times increase with increasing deposit height. A Caterpillar D8T bulldozer will be used to spread the tailings in layers approximately 1-m thick. As the ramp on the deposit will be shared with mining dump trucks of a much larger size, the filtered tailings contractor will also be responsible for sharing maintenance of the ramp in common use, using a grader, water truck, and backhoe.

16.7.10 Fuel

For the Phase 1 pre-strip (construction of haul roads), diesel fuel will be supplied by a major Peruvian fuel company, who will construct temporary fuel storage facilities (capacity 40,000 gallons) and provide a refuelling service in the field using fuel trucks (two on day shift and 1 on night shift).

For the Phase 2 pre-strip and the rest of the mine life, a fuel facility will be constructed near the main entrance gate. It will have 100,000 gallons of storage capacity (equivalent to 9 days of storage during the peak demand in 2021) and be equipped with two fuel bowsers for light equipment. The same company will be responsible for refuelling and lubrication of all mining equipment in the field, using two mixed fuel/lubrication trucks operating 24 h/day. Equipment from other areas will refuel at the fuel station also.

16.7.11 Explosives

For the Phase 1 pre-strip (construction of haul roads), Phase 2 pre-strip, and production years, a Peruvian explosives supplier will be responsible for constructing explosives storage facilities, supplying explosives, and providing a “down-hole” blasting service. Facilities to be provided are:

- Storage magazine for detonator
- Storage magazine for high explosives

- Roofed area for Ammonium Nitrate storage and loading silo
- Silos for emulsion storage

The storage magazine for detonators will be approximately 18 m^2 and will have sufficient capacity for 500 kg. The storage magazine for high explosives will be approximately 14 m^2 and have sufficient capacity for 1,800 kg. Boosters and detonator cord will be stored in this magazine. Both magazines will be separated by an earth wall, with a minimum distance of 19 m between the magazines.

The bulk emulsion storage silos will have storage for 160 tonnes distributed in two silos, which will be filled by the explosives supplier´s trucks which will bring the emulsion to the mine. These silos will be used to fill the mobile mixing unit trucks for use in the mine.

The ammonium nitrate storage shed will be a closed sided, steel-clad shed with an area of 375 m^2. It will have capacity for 300 tonnes of material in big bags of 1.00 or 1.25 tonnes.

All storage facilities will be surrounded by security fences and provided with external security lighting and lightning protection.

16.7.12 Truck Shop and Facilities

The truck shop will be located in the northwest part of the industrial area, with access from the mine passing by the primary crusher pad, and with access for light vehicles from the processing plant. The truck shop will include:

- Mine Operations and maintenance offices
- Toilets and changing rooms
- Satellite lunch room
- Deposit for consumables
- Area for lubricants and additives
- A total of 3 maintenance bays: (1) assigned to preventative maintenance, (1) assigned to corrective maintenance, (1) multipurpose.
- Light vehicle bays (2)
- Overhead bridge crane (40 tonne main, 10 tonne secondary)

A tire changing area of approximately 1,120 m^2, will be located next to the truck shop and includes a concrete slab for jacking trucks up and changing tires. In addition, there are two areas of approximately 500 m^2 for storage of tires. The area will have a container office and warehouse.

A truck and equipment wash area of 800 m^2 will be located next to the truck shop. It includes a concrete slab with steel protection rails, enclosures, and elevated platforms equipped with water cannons and hose reels.

16.7.13 Fleet Management System

As requested by Bear Creek Mining, a fleet management system has been included. The system considered is a Peruvian developed system and has been successfully implemented at several Peruvian mining operations. The supplier will be responsible for providing all hardware and software required and for operating the system for the first 5 years from the commencement of the phase 2 pre-strip (start of operations with mining equipment). From year 6 onwards, the system will be operated by the mining company's mine operations personal. The system is included in both the owner operator and mining contractor scenarios, as a mining company cost.

16.8 Work Schedule

Mining will be carried out using 8-m high benches, consistent with the NI43-101 report (M3 Engineering, 2015) and will intersect some historic underground workings. Mining will be carried out working 2 x 12-hour shifts per day, 365 days per year. In total, three crews of operations personnel will be required, one working day shift, one working night shift, and one on rest days. Crews will work a 7 day, 7 night, 7 day off roster. Maximum equipment working hours per day are estimated to be 20.50, resulting in a maximum possible utilization of 85.4%. Time lost on average per day is shown in Table 16-13.

Table 16-13: Calculation of Utilization

Description	Unit	Time
Hours per Day	h/day	24.00
Shift Change	h/day	1.00
Meals	h/day	2.00
Blasting	h/day	0.50
Training	h/day	0.00
Weather	h/day	0.00
Total Non-Utilizable Time	h/day	3.50
Total Non-Utilizable Time	h/day	20.50
Utilization (Max Possible)	%	85.4%

Personnel Requirements: Construction (Phase 1 Pre-strip and Construction of Haul Roads) are shown in Table 16-14.

Table 16-14: Pre-Strip Phase 1 and Construction of Haul Roads: Personnel

Personnel	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18	Jan-19	Feb-19	Mar-19	Apr-19	May-19	Jun-19	Jul-19
Foreman		3	3	3	3	3	2	2	2	1	1	
Operator		4	4	4	5	6	6	4	4	4	3	
Official		0	0	1	2	2	2	2	2	2	1	
Labourer / Helper		40	42	42	59	59	45	45	45	13	10	
Operator - Heavy Equipment		38	38	40	32	40	38	40	37	12	3	
Operator - Medium Equipment		84	84	84	92	113	105	105	64	12	6	
Maintenance		20	20	20	20	20	20	20	20	20	20	
Subtotal - Direct Labor		189	191	194	213	243	218	218	174	64	44	
Staff	8	19	19	19	19	19	19	19	19	19	19	8
Assistant / Helper	8	32	32	32	32	32	32	32	32	32	32	8
Subtotal - Indirect Labor	16	51	51	51	51	51	51	51	51	51	51	16
Contractor (Explosives)	19	19	19	19	19	19	19	19	19	19	19	
Total Personal	35	259	261	264	283	313	288	288	244	134	114	16

17 Recovery Methods

17.1 Site Layout Considerations

The Project site is located on steep sloping, high-altitude terrain that has limited flat space. These considerations required particular care in developing acceptable sites for the facilities.

The development of the site layout was based on maximizing the ease of operation and minimizing both capital and operating costs.

17.2 Process Description

The Corani Project process facility is based on a two stage sequential flotation concentrator generating two products: a lead concentrate enriched with silver and a zinc concentrate with slight silver presence.

The plant design incorporates the following process areas:

- Primary crushing and coarse ore Stock pile
- SAG Milling, followed by Ball Milling
- Selective Lead-Zinc flotation
- Concentrates Thickening and Filtering (Pb, Zn)
- Tailings Thickening and Filtering
- Dry Stack Tailings system
- Reagent preparation and ancillary facilities

17.2.1 Primary Crushing and Coarse Ore Stockpile

The primary crushing plant is designed to process 22,500 t/d of ore, on an operating schedule of 350 days per year, 24 hours per day at 70% effective availability (availability + Utilisation).

The ROM (run-of-mine) ore at, 1,000 mm maximum size, will be directly discharged onto a stationary grizzly assisted by one hydraulic rock breaker. The crusher feed hopper has a 240 t capacity, equivalent to 1.8 times a CAT 785 D 131 truck. The design allows sequential (not simultaneous) discharge on two different sides.

The design allows the hopper to provide gravitational feeding to the primary 50' x 65' gyratory crusher, in order to obtain a P80 = 150 mm size. The crusher product is unloaded on a series of conveyor belts carrying the material to the coarse ore stockpile. The design incorporates a dust suppression system.

The coarse ore stockpile (see Figure 17-1) having a live capacity of 11,250 tons and 44,530 tons of total capacity, with autonomy of 12 continuous working hours with respect to its treatment nominal capacity, and 2 days with respect to dead load, because the grinding area is operated with an availability of 92%.

Primary crushing product will discharge from the stockpile feed conveyor belt directly onto the coarse ore stockpile. Ore will be drawn from the stockpile by three reclaim apron feeders, two operating and one in standby. The reclaim apron feeders will discharge onto the stockpile reclaim/ SAG feed conveyor belt which will covey the material to the SAG mill. A belt scale, magnet and metal detector will be installed in the conveyor belt to prevent metallic elements reaching the mill. In this path there is also a ball feed system to feed 5 inches balls to the SAG mill.

The belt design incorporates a protective cover, reducing material losses due to wind action, or snow and/or rain reaching the belt.

17.2.2 Grinding and Classification Circuit Description

The grinding design has a nominal treatment of 22,500 t/d ore operating at 92% of availability. The final product of the grinding circuit has been determined from the metallurgy at 80% passing 106 microns (P80) at 32% w/w solids average concentration. The equipment has been designed for 90 microns.

The grinding circuit includes a semi-autogeneous mill at 9.1 m dia.x 4.3 m (EGL) with a 7,000 kW motor, provided with a trommel at the discharge end, operating in line and in open circuit with the second stage grinding ball mill at 7.3 m dia. x 11.9 m (EGL) with dual 7,000 kW motors, operating in closed circuit with hydrocyclones.

The design SAG-Ball Mill transfer size is 1.18 mm (T80). SAG Mill trommel oversize (pebbles), estimated at 25% of feed maximum will be recirculated to the mill via a pressurized water injection system called the Water Jet System. The trommel undersize and the ball mill product are both combined in the Sag Mill discharge pump box and pumped to the classification system with 10 hydrocyclones (2 standby) each 33 in. diameter. The combined mill discharges at 70% solids are diluted to 55% to feed the cyclones. The ball mill design includes a ball feeding system for 2" to 3" steel balls.

Further work will be required during detailed design on the pebbles handling system, once more data is obtained on their effects on SAG mill operation.

17.2.3 Lead Flotation

The hydrocyclone overflow flows by gravity to two conditioner tank, operating in series. In the first one zinc cyanide and sodium sulphide are added to depress pyrite and sphalerite, while in the second, lead activators and collectors, sodium isopropyl xanthate (SIPX) and promoter A-404 (AP 404) are added, as well as frother methyl isobutyl carbinol (MIBC). The conditioned slurry flows by gravity to the lead rougher flotation cells, consisting of 5 conventional forced air cells, 300 m^3 each in series.

In the event that the concentrate generated by the first two rougher cells is of high grade, the design of the plant has the flexibility (bypass) to direct the concentrate to the second cleaning stage, thus avoiding re-grinding and the first cleaning stages. The use of this option is highly recommended when working with minerals with fast flotation kinetics.

The concentrate obtained from the lead rougher flotation constitutes the fresh feed to the lead regrind circuit equipped with a vertical mill with 1500 kW, operating in open circuit with 8 hydrocyclones (2 standby) each 10" in diameter. The lead rougher concentrate is reground at 25-30 µm P80 and is pumped to the first cleaning flotation stage of 4 conventional cells in series of 50 m^3 each.

The lead first cleaner concentrate is pumped into the second cleaning stage (first column cell) of the flotation circuit. The first cleaner tails flow by gravity to the cleaner-scavenger circuit equipped with 4 cells x 50 m^3 cells working in series. The resulting concentrate is recirculated to the first cleaner feed box and tails are sent to the zinc flotation circuit.

The concentrate generated by the second cleaner is fed to the third cleaner flotation (second column cell). The tailings generated by the second lead cleaner flotation are recirculated to the first cleaning stage. The third cleaner concentrate (from 2nd column cell), becomes the final concentrate of the lead circuit, and feeds the lead concentrate thickener, while the lead third cleaner tailings are recirculated to the feed box of the second cleaner flotation (first column cell).

The flotation circuit was designed to send the tailings from the rougher flotation circuit and cleaner-scavenger flotation circuit to the tailings thickener, when the zinc flotation circuit is not in operating.

The design included a sump system to collect spills generated by the rougher and cleaner-scavenger flotation stages and recirculate them to the corresponding concentration stage. Provision is made for operational adjustments of pH and reagent and process water additions.

In general, the lead circuit has a robust design for handling of concentrates and tails generated by flotation. A pending engineering issue is related to ensuring that the maximum silver is recovered to the lead concentrate.

17.2.4 Zinc Flotation

Tailings from lead rougher flotation and the lead scavenger-cleaner circuits are pumped to two zinc circuit feed conditioning tanks operated in series. In the first, copper sulphate is added to activate the zinc while in the second a xanthate collector, frother and lime to adjust pH to a value around 11.

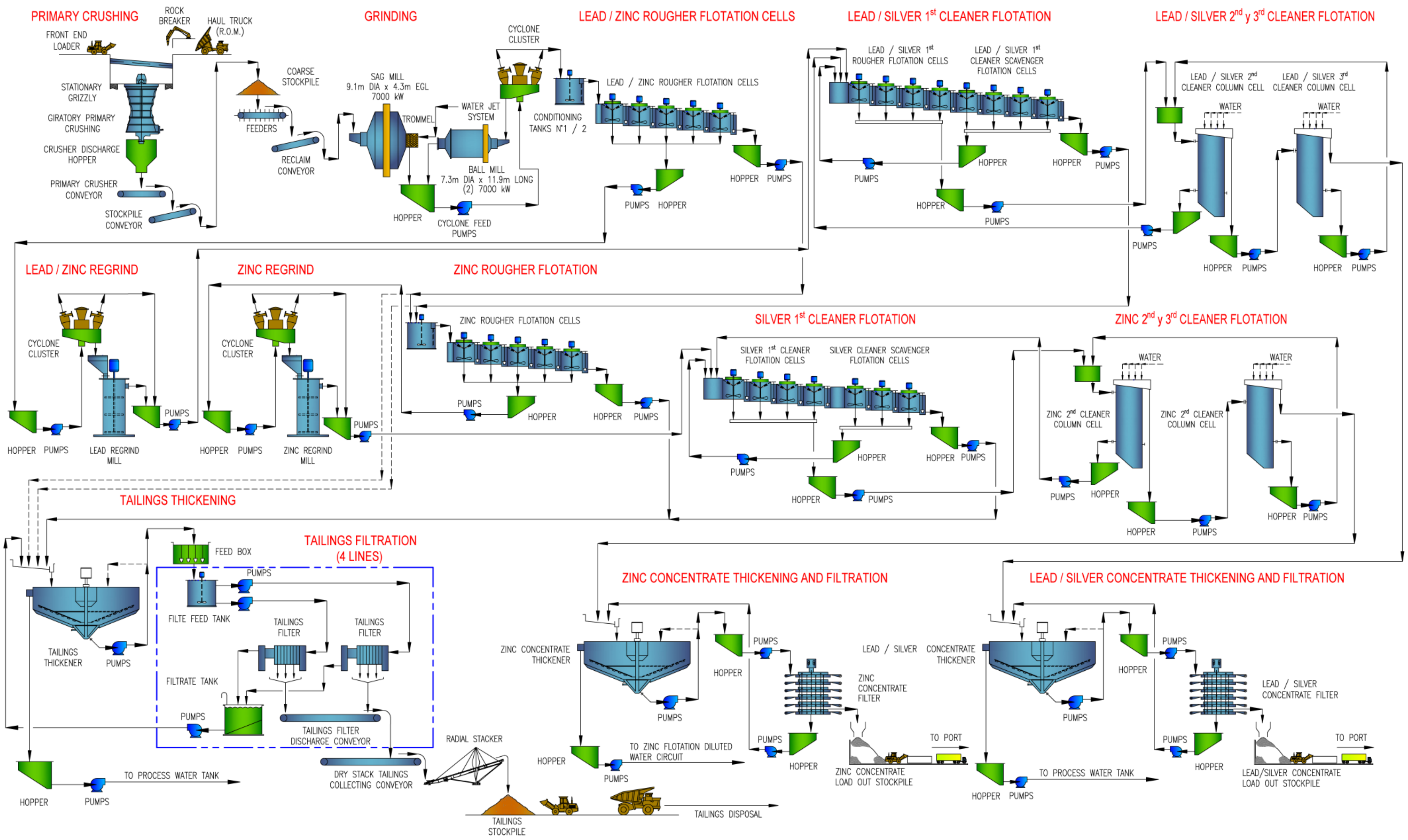
The conditioned slurry flows by gravity to the zinc rougher flotation with 5 conventional forced air cells each 300 m^3, operated in series The tailings generated by zinc rougher flotation feeds the tailings thickener, while concentrate is fed to the zinc regrind circuit equipped with a vertical mill with 1500 kW operating in an open circuit with 7 hydrocyclones (2 standby) each 10" in diameter. The zinc rougher concentrate is reground at 25-30 µm P80 and pumped to the zinc first cleaner flotation, 4 cells in series each of 40 m^3.

The zinc first cleaner concentrate is pumped to the second cleaner flotation (first column cell), while the first cleaner tailings flow by gravity to the cleaner-scavenger stage, of 3 cells in series, each 40 m^3. The zinc cleaner-scavenger concentrate is recirculated to the first cleaner feed, while the tailings are pumped to the final tailings thickener.

The concentrate obtained from the zinc second cleaner flotation (first column) is pumped to the second column cell for the third cleaner flotation stage. The second cleaner tailings are recirculated to first cleaner flotation, while the third cleaner concentrate becomes the final concentrate of the zinc circuit and is pumped to the zinc concentrate thickener. The third cleaner tailings (second column) are recirculated to the second cleaner flotation feed.

The overall flotation design includes on-line "sampling" diverting a slurry sample to the multi-stage on line analyser, in order to measure lead, zinc, iron and other elements which are necessary for control of the operation. The design included in the layout the necessary space to incorporate metallurgical sampler cutters to sample and assay the fresh feed to lead flotation, overall plant tailings and final lead and zinc concentrates. Figure 17-1 shows the general flow diagram of the Corani process plant.

Figure 17-1: Overall Process Flow Diagram for Corani Project



17.2.5 Lead Concentrate Thickening and Filtration

The lead concentrate will be dewatered in a 10 m diameter thickener, producing a 65% w/w solids average underflow, which is pumped into the lead concentrate filter feed tank. which has a residence time of 8.5 hours. This allows control of feed the filter as required to produce a final concentrate with 8% average moisture.

The thickened lead concentrate will be pumped to the lead concentrate automatic vertical pressure filter Larox PF12, including 20 plates of 900 mm x 1750 mm each, with own PLC to control and automate the feeding, pressing, blowing and filtrated cake unloading stages, as well as cloths washing or any other activity required for operations. Filter cake will discharge to a covered lead concentrate stockpile.

The lead thickener overflow is pumped to the process water tank. The thickener also receives water coming from the concentrate filtration stage and spills generated by the area.

17.2.6 Zinc Concentrate Thickening and Filtration

The zinc concentrate will be dewatered in an 8 meter diameter thickener, producing an average underflow discharge at 65% w/w solids which is pumped to a 9,8 hour residence time filter feed tank This allows regulation of the feed to the filter, as required to produce a final concentrate at 8% average moisture.

The thickened zinc concentrate will be pumped to the zinc concentrate automatic vertical pressure filter Larox PF12, including 16 plates each 900mm x 1750mm. The filter has its own PLC to control and automate feeding, pressing, blowing and filter cake discharge stages, as well as cloth washing or other activity required by operations. Filter cake will discharge to a covered zinc concentrate stockpile.

The zinc thickener overflow is recirculated to the zinc flotation circuit. The thickener also receives filtrate from the concentrate filtration stage, and from potential spills generated in the area for solids recovery.

17.2.7 Tailings Plant Thickening and Filtering

Tailings from the zinc flotation circuit are dewatered in a 50 m diameter thickener producing and the underflow containing 45-55% solids w/w is pumped to the tailings filtration section. The underflow is pumped to 4 filter feed tanks. An additional feed tank will be added in later years to accommodate increased tailings flows.

Each feed tank feeds directly to two filters via centrifugal pumps. In emergency cases, the thickener may discharge all by gravity to the emergency pond.

Tailings dewatering is carried out by eight (plus 2 in future) Larox FFP 3512 plate horizontal press filters, including 74 cameras with 3,500 mm x 2,500 mm plates each. The thickened tailings are gradually fed onto the press filters during the corresponding filtering cycle, allowing to obtain a filtered tailing with 17% w/w moisture.

The filtrate and wash water will flow by gravity to the tailings filtrate tank, which will then be pumped to the tailing thickener.

Each filter has its own PLC to control and automate feeding, pressing, blowing and filtered cake discharge stages, as well as cloth washing and any other activity required by operations.

17.2.8 Tailings Disposal

Filters will discharge the 17% w/w average moisture tailings by gravity onto 4 belt conveyors (one for each 2 filters) that transfer these tailings to a belt conveyor provided with a belt scale for operational control. This in turn discharges via a radial stacker to a stockpile. From the stockpile, a front and loader will load dump trucks~~,~~ and until year 8 of operation will be transported to the Waste Rock and Dry Tailings Storage area [(1)] located at the south of the Main pit. Tailings will be co-disposed with the waste rock generated by the mine. Between years 9 and 11 of the mine's life, the tailings will be transported to backfill mined out pits at the mine. From year 12 onwards, the tailings will be again be deposited at the Waste Rock and Dry Tailings Storage area until the end of the mining operation.

For years 12 to 18 Bear Creek Mining has postponed their decision, either to continue using their truck fleet, or to use conveyor belts.

17.3 Process Design Criteria

The main process design criteria [(2)] used in the development of Phase 1 FEED Engineering of Corani Project are described below.

17.3.1 Metallurgical Testing

No additional metallurgical tests were carried out during the phase 1 FEED engineering.

The metallurgical test campaigns completed from year 2006 to 2015 have produced variable recoveries of the Project metals of interest (Ag, Pb and Zn).

From the observed trends during the lock cycle tests performed between years 2008 and 2009 statistical analysis was performed on the recoveries generated by element of interest (Pb, Ag, Zn), in order to identify the metallurgical behaviour of the samples studied. [(3)(4)].

Parameters that influence recoveries are:

For Lead Recovery:

- Zinc Feed Grade
- Mine Elevation
- Galena Present (%) vs. Lead Phosphates
- Amount of oxides present (Fe, MnO)

For Zinc recovery:

- Zinc Feed Grade
- Mine Elevation
- Pyrite Feed Grade

More information about this can be found in Chapter 13 of this report.

1 Depósito de Desmontes de Mina y Relaves Principal (DDMRP).

2 “Process Design Criteria”, GMI, document N° 161655-000-0-CD-0001_0, May 2017

3 “The Recovery of Silver, Lead and Zinc from Corani Sampler”, SGS Minerals Canada, document N° 50000-004, January 2008

4 “The Mineralogy and Flotation of Samples from Corani Deposit”, SGS Minerals Canada, document N° 50000-006, November 2009

17.3.2 Process Design Criteria

A process workshop with Bear Creek specialists was held in February 2016 where it was agreed to use the following design criteria for the development of the FEED phase 1 engineering. It should be noticed that the last mining plan was approved in June 2017:

Table 17-1: Total Tons and Average Grades pondered per ore type ([5])

	2018-T1	2018-T2	2018-T3	2018-T4	2019-T1	2019-T2	2019-T3	2019-T4	2020-T1	2020-T2	2020-T3	2020-T4	2021-T1	2021-T2	2021-T3	2021-T4	2022-T1
Tonnage. Kt	413	1.181	1.733	1.969	1.969	1.969	1.955	1.969	1.969	1.969	1.969	1.969	1.969	1.969	1.969	1.969	1.964
Grade Ag, g/t	148	114	97	85	90	86	84	78	86	56	44	94	80	73	88	68	67
Grade Pb, %	1.26	1.25	0.98	1.04	1.04	1.04	1.2	1.15	1.12	1.1	0.9	1.49	1.11	0.98	1.12	1.14	1.35
Grade Zn, %	0.5	0.81	0.84	0.77	0.82	0.72	0.7	0.73	0.63	0.88	0.9	0.83	0.66	0.48	0.93	0.82	0.69

	2022-T2	2022-T3	2022-T4	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	Weighted
Tonnage, kt	1.969	1.969	1.969	7.876	7.876	7.876	7.876	7.876	7.876	7.876	7.876	7.876	7.876	7.876	7.870	7.463	
Grade Ag, g/t	56	56	63	60	35	49	56	50	47	49	46	34	38	36	31	29	52
Grade Pb, %	0,87	0,89	0,97	0,97	1,03	0,97	0,96	0,76	0,78	0,82	0,9	0,89	0,76	0,93	0,72	0,57	0,92
Grade Zn, %	0,49	0,74	0,88	1,09	0,58	0,33	0,35	0,34	0,39	0,38	0,44	0,48	0,44	0,39	0,61	0,78	0,57

It is important to indicate that during the development of the Project optimization of the mining plan continued, generating a final version after the end of the Project. An analysis of the modifications can be found in Chapter **¡Error! No se encuentra el origen de la referencia.** of this report. More up-to-date information can be found in Chapters 15 and 16 of this report.

The process design assumed these overall recoveries for lead, zinc and silver per concentrate (lead and zinc concentrates).

Table 17-2: Global recoveries per concentrate type ([6])

Concentrate	Lead, %	Zinc, %	Silver, %
Lead Concentrate	61.15	---	50.42
Zinc Concentrate	---	66.97	20.33
Total	61.15	66.97	70.75

These recoveries were used only to generate the mass balance and not for calculation of metal recoveries in the financial model, where the 2015 report values were assumed to be correct.

The following tables present the main process design criteria used for the phase 1 of the detail engineering.

5 Based on file "Corani Mining Plan, Mine Plan Case C W Bofedal.xlsx", data available to the date of the design.

6 Delivered by BCM

Table 17-3: Comminution Testwork parameters

	Value	Unit
Crushing Work Index	8.1	kWh/t
Drop Weight Index (DWi)	3.01	kWh/m^3
Autogeneous Grinding (Mia)	11.96	kWh/t
Roll crushing (Mih)	7.58	kWh/t
Crushing (Mic)	3.98	kWh/t

Table 17-4: SAG modelling parameters

	Value	Unit
A	60.33	Percentil 15
B	1.17	Percentil 15
Axb	75.91	Percentil 15
Ta	0.76	Percentil 10

Table 17-5: Mine Production Schedule

	Value	Unit
Life of Mine	18	year
Production rate	7,875,000	tpy
Days of Operation per year	350	d
Production rate	22,500	tpd

Table 17-6: Primary Crushing and Crushed Ore conveyor Design Criteria

	Value	Unit
Production rate	22,500	tpd
Availability	70	%
Ore Crushing rate, Design	1,339	tph
Crushing Feed F100	1,000	mm
Crushing Feed F80	460	mm
Crushing Product P80	150	mm
Mine Truck	135	t
Downloading Point	2	not simultaneous
Moisture	3	%

Figure 17-2: Distribution of ROM particles size at

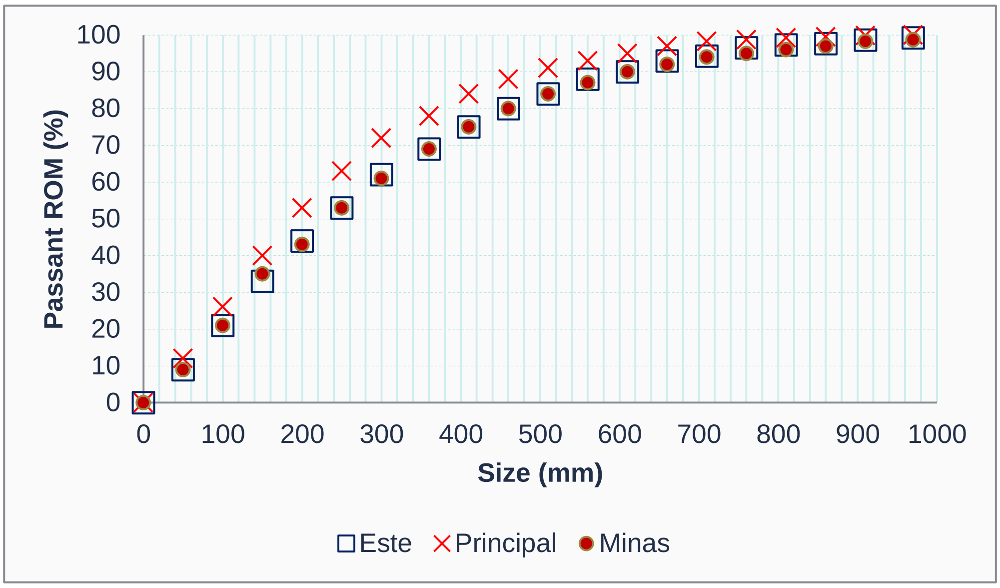


Table 17-7: Crushed Ore Stockpile Design Criteria

	Value	Unit
Total Capacity	44,530	t
Live Capacity	11,250	t
Feeders	3	units
Moisture	4	%

Table 17-8: SAG Mill Design Criteria

	Value	Unit
Production rate	22,500	t/d
Availability	92	%
Feed F80	150	mm
Product T80	1,180	µm
SAG Mill feed rate	1,019	t/h
Critical Speed	75	%
Circuit, type	open	---
Density	75	%
Pebble Generation, max	25	%
Pebble recycling system	Water Jet	---

Table 17-9: Ball Mill Design Criteria

	Value	Unit
Circuit Type	Closed	
Critical Speed	78	%
Availability	92	%
Fresh Feed	1,019	t/h
Feed F80	2,700	µm
Product P80	90	µm
Slurry Density	70	% w/w
Ball Load	37	%

Table 17-10: Hydrocyclones Design Criteria

	Value	Unit
Number (operating/stand-by)	8 / 2	Units
Circulating Load	350	%
Overflow	32	%
Cyclones O/F P80	90	µm

Table 17-11: Lead Flotation Design Criteria

	Laboratory	Factor	Holding up
Rougher	12 min	2.5	15%
First Cleaner	5 min	2.5	10%
Cleaner-Scavenger	6 min	2.5	10%

Table 17-12: Zinc Flotation Design Criteria

	Laboratory	Factor	Holding up
Rougher	12 min	2.2	15%
First Cleaning	6 min	2.0	10%
Scavenger	5 min	2.0	10%

Table 17-13: Column Cells Design Criteria

	Lead Flotation	Zinc Flotation	Unit
Slurry Speed	10 – 48	10 - 48	m/h
Froth Carry	2 – 4	2 - 4	g/min/m^2
Lifting Capacity	2	2	t $_{conc}$/h/m^2
Number of Cells	1	1	
pH	6 - 7	11,5	

Table 17-14: Regrinding Design Criteria

	Lead Flotation	Zinc Flotation	Unit
Circuit Type	Open	Open	
Quantity	1	1	Units
Type of Mill	Vertical	Vertical	
Feed, F80	55	55	µm
Product, P80	25 - 30	25 - 30	µm
Comminution Specific Energy	17.9	30	kWh/t

Table 17-15: Concentrates Thickening Design Criteria

	Lead	Zinc	Unit
Quantity	1	1	
Type	High Rate	High Rate	
Specific Capacity	1.1 (7)	0,7 (7)	$m^3/m^2/h$
pH	6 – 7	11	
Slurry Density (U/F)	65	65	% w/w

Table 17-16: Tailings Thickening Design Criteria

	Value	Unit
Quantity	1	
Type	High Rate	
Specific Capacity	0.5	$t/h/m^2$
pH	11	
Pulp Density (U/F)	55 - 60	% w/w

Table 17-17: Concentrates Filtration Design Criteria

	Lead	Zinc	Unit
Head Tank	1	1	Units
Residence Time	8.5	9.8	H
Filters Quantity	1	1	Units
Type	Vertical High Pressure	Vertical High Pressure	
Specific Capacity	0.566 (8)	0.456 (8)	$t/h/m^2$
pH	6	12	
Pulp Density	65	65	% w/w
Final Moisture	8	8	% w/w

7 from feasibility study (M3)

8 data from vendor (Outotec)

Table 17-18: Tailings Filtration Design Criteria

Tailings Filtration	Value	Unit
Head Tank	4	units
Residence Time	1.5	h
Filters Quantity	8	Units
Type	Horizontal High Pressure	
Specific Capacity	0.145 [9]	t/h/m^2
pH	11	
Pulp Density	55	% w/w
Moisture	17	% w/w
Tailings filtrate tank	1	units
Residence Time	18	min

Table 17-19: Lime Consumption and Design Criteria

Quicklime	Value	Unit
Type	Bulk	
Storage Type	Bin	
Grade	>65	%CaO free
Specific Consumption	3.5	kg/t
Type of Plant	Vertical Mill	
Concentration	20	% w/w

Table 17-20: Other Reagents Consumption

Reagents	Value	Unit
Sodium Isopropyl Xanthate (SIPX)	40	g/t
A404	15	g/t
Zinc Sulfate	620	g/t
Sodium Cyanide	210	g/t
Copper Sulfate	290	g/t
Methyl Isobutyl Carbinol (MIBC)	50	g/t
Sodium Sulfite	505	g/t
Sodium Hydroxide	10	g/t
Flocculant	20	g/t
Antiscalant	5	g/t

9 data from vendor (Outotec)

Table 17-21: Consumables [10]

Consumables	Value	Unit
Primary Crusher Liners	8	g/t
SAG Mill Liners	50	g/t
SAG Mill Balls (5")	500	g/t
Ball Mill Liners	30	g/t
Ball Mill Balls (2" – 3")	500	g/t
Regrind Balls	20	g/t
Regrind Mill Liners	10	g/t

Table 17-22: Samplers Type - Online

	Type	Use	Quantity
Lead Flotation Feed	On Pressure	Operation	1
Lead Rougher Concentrate	On Pressure	Operation	1
Lead Regrinding	On Pressure	Operation	1
Lead Rougher Tailings	On Pressure	Operation	1
Lead Cleaner-Scavenger Tailings	On Pressure	Operation	1
Final Lead Concentrate	On Pressure	Operation	1
Zinc Rougher Concentrate	On Pressure	Operation	1
Zinc Regrinding	On Pressure	Operation	1
Zinc Rougher Tailing	On Pressure	Operation	1
Zinc Cleaner-Scavenger Tailings	On Pressure	Operation	1
Final Zinc Concentrate	Rotating	Operation	1
Final Tailing Plant	On Pressure	Operation	1

In addition, Sedgman has recommended including metallurgical grade sample cutters in order to ensure the results against internal/external audits, in particular for the following operational points:

- Fresh Feed to Rougher Lead Flotation
- Final Lead Concentrate
- Feed to Rougher Zinc Floatation
- Final Zinc Concentrate
- Final Plant Tailings

Sufficient space and height were included in the design to include this equipment during the next stage of the Project.

10 data supplied by BCM for Opex Estimation

17.4 Water Requirement

The engineering design has considered recycling the process water recovered from the dewatering system to avoid any release of water in the environment, and at the same time prevent negative impacts on the efficiency of its operational units.

The site-wide water balance (see Section 20) has determined that the plant water requirement can be met even in extreme dry conditions. In addition, the process can consume the mine contact water, thus preventing the need for environmental discharge or water treatment.

The fresh water demand per area can be seen in Table 17-23:

Table 17-23: Fresh Water Consumption by Area

	m^3/d	l/s	m^3/t
Mine (Dust suppression)	488	5.65	0.022
Crushing and Coarse Ore Stockpile (Dust suppression)	488	5.65	0.022
Grinding	151	1.75	0.007
Flotation, Regrinding and Concentrates Thickening and Filtration	2328	26.94	0.103
Tailings Thickening and Filtration	65	0.75	0.003
Reagents	830	9.61	0.037
Potable Water	90	1.04	0.004
TOTAL	**4,441**	**51.4**	**0.197**

Table 17-24 shows the process water consumption by area.

Table 17-24: Process Water Consumption by Area

	m^3/d	l/s	m^3/t
Grinding	46,824	541.9	2.081
Flotation	11,821	136.8	0.525
Concentrates Filtration	43	0.5	0.002
Tailings Filtration	11,027	127.6	0.490
Flocculant Dilution	1,500	17.4	0.067
Miscellaneous plant	52	0.6	0.002
TOTAL	**71,268**	**824.9**	**3.167**

The proposed plant design allows the flexible management of recovered water and optimizes the distribution according to the recovered water source.

Based on Project specific information, recovered water is expected to contain dissolved metal Cu^{+2}, Pb^{+2}, Zn^{+2}, Fe^{+2}, Fe^{+3}, Ag^{+}, colloidal precipitate, recycled organic reagents and derived products due to degradation/oxidation of the same (lime, collector, depressor, dispersant, flocculants), compounds generated by reduced sulphur (RSC) - such as SO_3^{2}-, $S_2O_5^{2}$- and $S_4O_6^{2}$-, hardness (Ca^{+2}, Mg^{+2}),dissolved and suspended solids (TDS and TSS), organic carbon (TOC) and dissolved contaminants.

As an example in lead (galena flotation, PbS), metal ions such as Cu^{+2}, Pb^{+2}, Ag^{+} and Fe^{+2} which should be present in the recirculated water, could involuntarily activate any sphalerite (ZnS) surfaces, altering the metallurgical behaviour. Another effect is related to the reagents to be used in each of the flotation stages, especially if at the time the preparation is not carefully controlled.

17.4.1 Water Distribution Strategy

Fresh water will come from the non-contact water conveyance system within basin that contains the project (see Section 18). It is expected to be good quality.

Engineering has considered reusing 100% of the water recovered from the process, which is considered suitable for use if some basic distribution criteria are applied. However, the mine will be preferentially sourced from the contact water pond. This water may have elevated salt, dissolved metals, and acidity concentrations. Despite these concerns, it is considered suitable. When necessary (when the contact pond is dry, or for applications requiring better water quality), water will be collected from the non-contact water pond. The drainage basin near the mine has sufficient water to supply operations (see Section 20).

In the attached table a summary of water consumption and distribution during the process is shown:

Table 17-25: Water Balance for Process Plant

	m^3/d	l/s	m^3/t
Ore Moisture	696	8.1	0.031
Make up Fresh Water	3,851	44.6	0.171
Water Losses in Concentrates	48	0.6	0.002
Water Losses in Tailing	4,499	52.1	0.200
Total Water Recovery	71,268	824.9	3.167

The Corani plant has two flotation circuits (Lead and Zinc) working in series but demanding different operational conditions. While the lead circuit should work with a pH between 6 and 7, the zinc should operate at a pH 11.

The required reagents per type of flotation are shown in the table below.

Table 17-26: Flotation Reagents by Circuit

	Lead	Zinc	Observation
Zinc Cyanide	Yes	--	Zinc depressant
Sodium Sulphite	Yes	--	Zinc depressant
A404	Yes	--	Lead promotor, no water needed
Isopropyl Sodium Xanthate	Yes	Yes	Collector
Flocculant	Yes	Yes	
MIBC	Yes	Yes	No water needed
Copper Sulphate	---	Yes	Zinc Activator
MIBC	Yes	Yes	No water needed
Lime	---	Yes	pH Modifier

Water recovered from the lead thickener and lead concentrate filter will have high concentrations of Zn depressants with a pH slightly higher than 7, making it very attractive for reuse in its lead flotation and the design incorporates this re-use with appropriate controls.

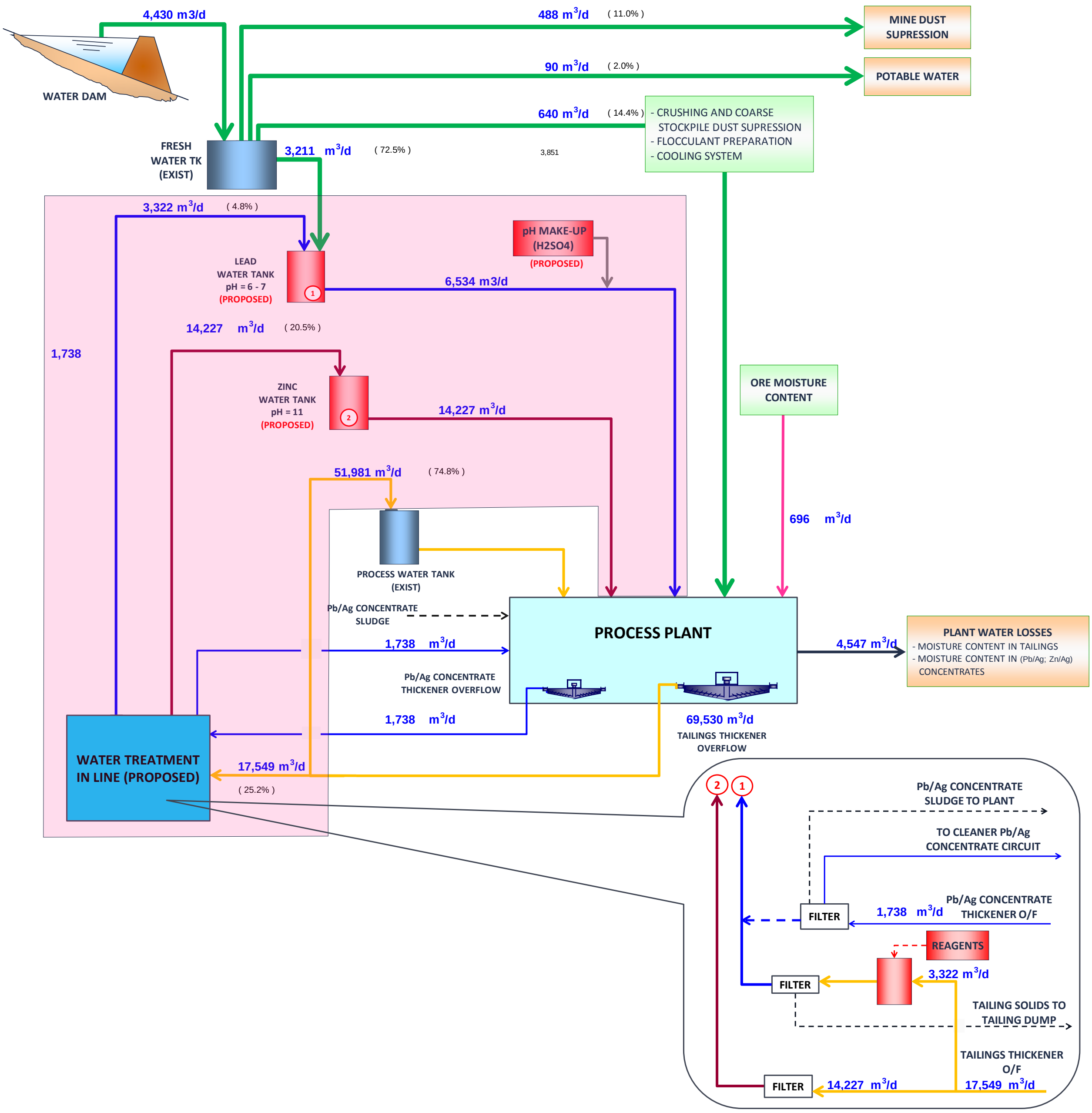
It is also estimated that from the dewatering stages of the Pb/Ag and Zn/Ag concentrates and due to the high content of silver, it is possible that thickener overflows contain significant fine solids. Inline filters are incorporated to recover these solids.

Similar conditions occur in the zinc circuit which must operate at pH 11 to optimize selectivity within the floatation circuit. However, this condition also makes it less advisable of being used directly within the lead circuit, but instead it can be used in the milling stage because of its high pH.

At the end of the engineering an operating philosophy was developed to recycle and reuse the water in the process plant, depending on its origin.

Figure 17 3 shows a schematic summary of the study recommendations.

Figure 17-3: Water Distribution and Treatment Strategy - Proposed



: Additional equipment for water treatment

: Fresh water

: Treated Water pH = 6 – 7

: Treated Water pH = 11

: Process Water no treated from lead concentrate thickener and tailings thickener

: Water content in fresh Ore

The recovered water to be used must be pre-treated with Hydrogen Peroxide and Activated Carbon in solution and pass through in-line filter units in order to allow the extraction of any colloidal fine particles and free radicals. This treatment will be necessary since the lead circuit is not able to generate the required water for recycling.

The recovered water to be used must be pre-treated with Hydrogen Peroxide and Activated Carbon in solution and pass through in-line filter units in order to allow the extraction of fine particles and free radicals. The zinc circuit does not require treatment of its reclaimed water because it comes from its own tailings thickener, and that could also be used in the milling circuit due to its high pH (11) which helps to reduce wearing due to the steels corrosion involved, such as balls and mill linings. .

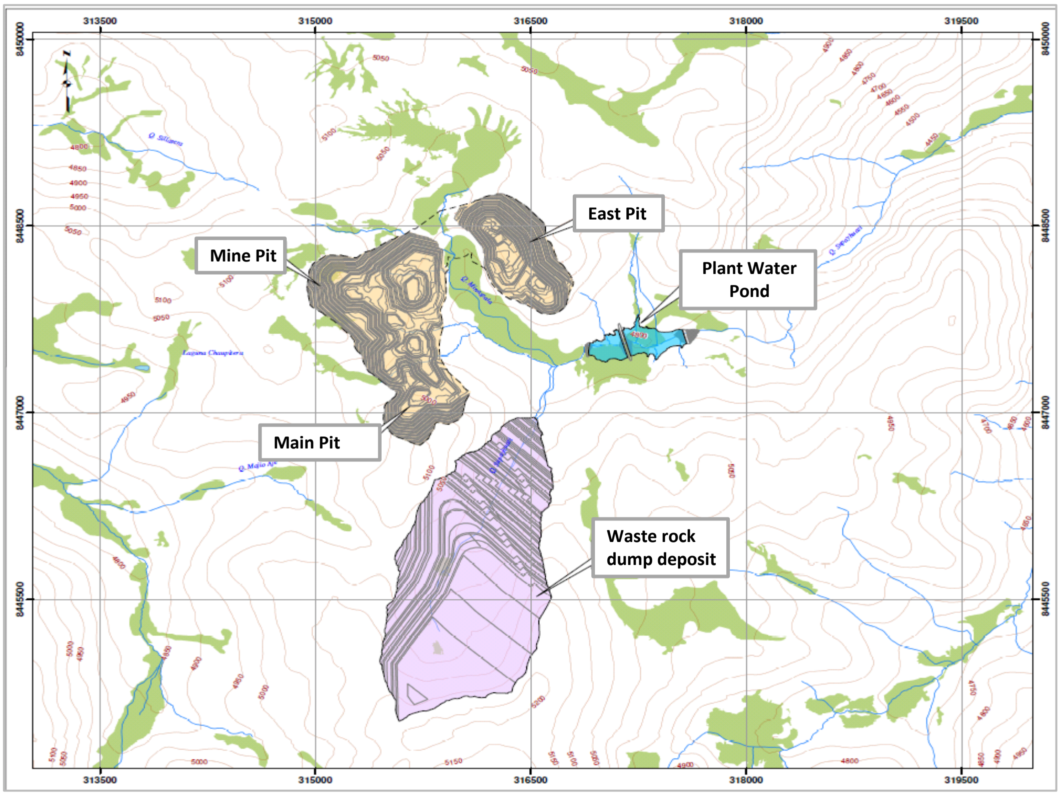
17.4.2 Fresh Water

The Project contact and non-contact waters will be managed in Plant Water Pond.

The pond has a storage capacity of 782,000 m3 and is located northeast of the process plant, downstream of the mine and DDMRP, of the borrowing material areas, and process plant (see Figure 17-4). Its design includes two dams for operation. One that separates the Contact Water from the Non-Contact Water, and one that separates the non-contact water from the downstream zones.

The justification for the pond size and further details on the water balance can be found in Section 20. .

Figure 17-4: Water Plant Pond location



Its design makes it possible to handle the contact water runoff and meet the water demand generated by the plant during the driest periods of the year (from May to September). During this time, it is also considered to release an ecological flow of 2.6 L/s rate, according to the commitment in the Environmental Impact Study.

17.5 Power Consumption

The mineral processing plant has been classified in 8 main areas, which incorporate mechanical and electrical equipment, requiring power for its operation. The average annual energy consumption amounts to 332,894 kWh/year, which corresponds to a specific consumption of 42.27 kWh per ton of processed ore.

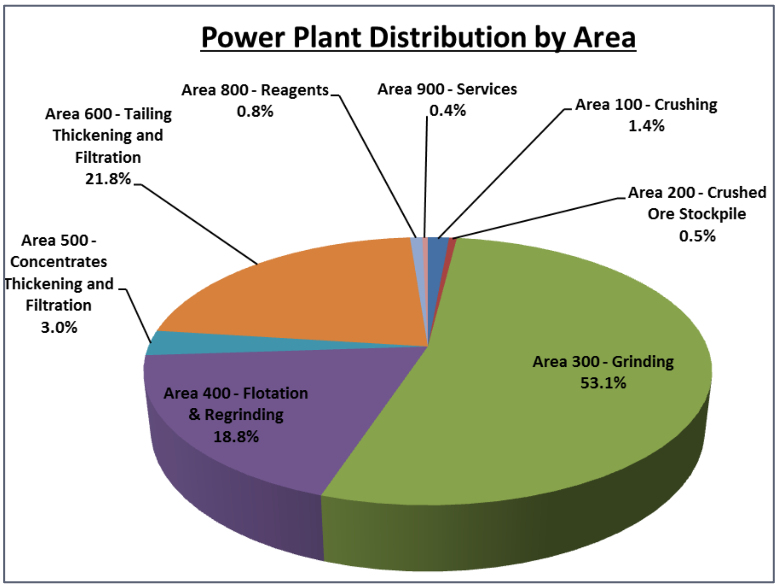
Table 17-27: Process Plant Average Power Consumption by Area [(11)]

Average power (MWh/year)	
AREA	**GMI FEED**
Area 100 – Crushing	4,804
Area 200 – Crushed ore Stockpile	1,748
Area 300 – Grinding	176,903
Area 400 – Flotation & Regrinding	62,688
Area 500 – Concentrates Thickening and Filtration	10,143
Area 600 – Tailings Thickening and Filtration	72,581
Area 800 – Reagents	2,786
Area 900 – Services	1,241
TOTAL	**332,894**

As is shown in Figure 17-5, the areas of highest energy consumption corresponds to area 300, Milling and Classification (53.1%), followed by Area 600 Tailings Thickener and Filtering (21.8%) and Lead and Zinc concentrates Flotation Area (18.8%), while the remaining areas of the plant as a whole account for 6.3%.

[11] from Operating Cost Estimation document N°161655-000-0-ECO-0002 Rev 1, GMI.

Figure 17-5: Process Plant Power Distribution by Area



17.6 Control System

The control is based on a process automation system and a set of controllers and PLCs with modular and open systems, based on recognized and tested standards of similar processing plants (SQL, HART, Ethernet, TCP/IP, Profibus DP and PA, IEC-61131-3).

In order to operate efficiently and safely, the Corani plant will be equipped to monitor most of the process variables from the control room, facilitating self-diagnosis and failure reporting and immediate alarm management.

The design considered distributed type control architecture (client/server), with disseminated controllers in electrical rooms and remote Input/Output cabinets (RIO's) at the field. The control cabinets will be responsible for controlling the process in each area of the plant.

Critical process controllers have a redundant processor and power supply. The control system will communicate with the servers and workstations of the main control cabinets located in the electrical rooms through a redundant Ethernet network, while the communication between the control cabinets and RIO's will be done through a Profibus DP fieldbus. For both cases, communication is via multimode optic fibre.

Regarding field instrumentation, the controller’s action, alarm switches and final control devices will be fail-safe, while the transmitters to be used would have a local or remote indicator. Conveyor belts will have pull cords and compact misalignment sensors, which are robust, safe and easily maintained. The installation of all instruments is at floor level, on access platforms, or operations stairways.

The automation system will also allow control of the processing plant from the main control room, which is located in the milling installation, in order to monitor and control process equipment, motors, screens, control loops and interlocks, and register operator inputs and diagnostics, in order to take preventive and corrective actions regarding tonnage, reagent dosage, pond levels, alarms and other parameters in the field.

Critical controllers will have redundant power supply sources.

17.6.1 Process Control Philosophy

The ore (run-of-mine) will arrive the crushing area in dump trucks. At the unloading area there are 2 traffic lights supervising the trucks unloading, ensuring alternate unloading (not simultaneous), Dumping permission also depends on the state of the crusher, since this equipment must be in operation and free of process and safety alarms. The operation of traffic lights is manual and automatic; in automatic mode, control will be supervised by the central PLC and, depending on the state of sensors reflecting the presence of trucks in the ramps, the state of the crusher and the hopper level sensor, the logic will provide permission to unload. While in manual mode, the decision to permit discharge will be made by the operator from the control room from the SCADA system.

Operational continuity will depend on the fact that there is no ore bridging in hoppers, damage or misalignment, of conveyor belts, among other variables. These actions will be monitored and directed permanently from the main control room.

The grinding circuit will work directly with the information generated by a pressure sensor located in the overflow line of the classification battery, in order to control the P80 (about 106 microns) required for the flotation stage.

The control logic establishes a control loop of the nested type, which allows balancing the pulp level in the mill pump box, the cyclone classification pressure, the circuit internal water balance, the product size (P80) and the flow of fresh ore feeding the mill.

Operational control of the flotation area will be supported mainly by the information generated by the X-ray analyser which will report the element of interest grades for each of the flotation stages, which might indirectly act on the reagents addition, (lime, foams, collectors or depressants), or facilitating the actions (local route) on the foam height inside the cells in order to individually regulate the flotation velocity.

The grade analyser may also report to operations if the zinc grade is low enough to make the decision to bypass the lead flotation tailings directly to the final tailings thickener or to maintain the operation.

At the lead and zinc concentrate thickening area, operations control will be through densimeters and flowmeters on the thickeners discharge, in order to generate the solids concentration required by the filters. At a second control level, the area design also allows for recirculation of thickened pulp if its density is less than that established in the design criterion, as set by operations.

In general, the concentrate filters will be controlled by their own PLC and monitored from the plant control room. In addition, each filter will work with its own programming facilitating the generation of filtering cycles previously agreed with the equipment supplier, in order to ensure the treatment rate ($t/m^2/h$) established according to design.

The control system for the thickener-tank system feeding filters and final tailings filters will be similar to that for concentrates. In case the final concentrate moisture is higher than the expected (about 20% w/w), the design incorporates reversible belts allowing discharge to the floor for re-handling, which will facilitate operation during plant commissioning and start up.

18 Project Infrastructure

The infrastructure for the Corani Project requires significant development and planning. The site is remote, at high elevation, and a considerable distance from major urban areas. The infrastructure developed for the Project includes transportation, process buildings and related facilities, water supply and management, power supply, communications, and material storage stockpiles.

Several project components were optimized subsequent to the July 2015 Technical Report by M3 Engineering (M3 PN 140135, 17 July 2015). Detailed engineering studies, site investigation work, and laboratory testing programs were ongoing at the time of the July 2015 report. The optimizations were presented in December 2015. Several significant changes have been made in the design approach in order streamline the Project and minimize Capital and Operating costs. The most significant changes have been advanced through additional fieldwork and detailed engineering to support the optimization concepts presented in this study. A summary of the infrastructure related work performed subsequent to the 2015 report for the present 2017 study is presented below.

- More precise and geo-referenced topography completed
- Soil mechanics studies completed
- Bridge and tunnel designs for the main project access road reviewed and in some cases revised
- Review and optimization of the overall Project layout
- Review and update of Project hydrology and hydrogeology
- Evaluation and development of Project accesses
- Development of accesses for mine vehicles (Haul Roads)

The current components and arrangement of facilities is described in this section of the Report. Several of these project components are described in more detail in other sections of this report, and only a general description of the relevant aspects of the project infrastructure-related components is given here. The following components of the project are described in the sub-sections that follow

- Transportation – Access, and Site Roads
- Site Buildings and Facilities
  - Mine Services Facilities
  - Administration Facilities
  - Process Facilities
  - Camp Facilities
  - Water Supply and Management Facilities
  - Power Supply and Distribution
  - Communications Systems
  - Waste Disposal Facilities
  - Waste Rock and Tailings Management Facilities

18.1 Transportation

Transportation to and around the site is by roadways that have been developed and improved to accommodate the demands of the project. An access road has been designed to link the project site to the Interoceanic Highway that provides access to the town of Macusani and to the rest of the country for receiving supplies and delivering products. The lead and zinc concentrate produced by the mining and mineral processing operations will be delivered to the Port of Matarani or other destination via trucks using the access road and Peru's public highway system.

18.1.1 Access Road

The Main Access Road to the Corani Project, as designed by GMI and Anddes Asociados SAC (Anddes), will be a new 46 km highway connecting to the existing Interoceanic Highway (34B) which is in turn connected through the existing Peruvian highway system to the Port of Matarani, 632 km. from the Project site. The Port has facilities for concentrate shipment. Figure 18-1 shows the Plant access and site roads. The Interoceanic Highway is a two-lane, paved highway that connects the Peruvian port cities of Matarani and Ilo.

The access road intersects the Interoceanic Highway at km 198+ 500 just before it crosses the Huiquisa River Bridge, about 16 km north of Macusani. The road has three Sections. Section 1 begins at the Highway as a new road, following the left bank of the Macusani River and improving highway existing P500 with new stretches to bypass several populated centres, improving highway safety. At the beginning of 2017, GMI carried out field investigations which identified unfavourable conditions along the routing of Section 1 used in previous studies.

The studies showed the following

- Tunnels 1, 2, 4 and 5, were covered by less than three diameters of the tunnel cross section (21 metres) which, given the poor quality of the rock discovered in the fieldwork, would not guarantee the stability of the proposed tunnel.
- Strong alteration of the rock and rock falls at both tunnel entrances.
- Chillcuna ravine shows a cover of residual and alluvial soils of fine and loose loamy sand, which are easily eroded and unstable due to their poor cohesion. This had produced landslides along more than 500 meters of the right margin of this ravine. In the case of the exit portal (1 + 660), slides were identified on the lower slope for almost 500 meters on the right bank of the ravine, which would require stabilization and containment works on the lower slope as well as consideration of using false tunnels at both portals.
- Geophysical tests indicate that the depth of loose sandy soils at the site of the bridges is between 15 and 24 meters which required moving the bridge to a location with better foundation conditions and to avoid areas of unstable sandy slopes.

The Access road design has been re-routed to avoid these unfavourable sections.

On rough and rugged terrain, for reasons of economy and compatibility with the category of road, it will follow the natural undulations of the terrain without compromising sightlines and safety features. There are circumstances that will require using "U" or switchback curves, as is the case in very tight ravines and sectors of slope development, so that in these sectors there will be occasional reductions of the posted speed to maintain a reasonable project cost. This will also require an increase in a few slopes in some sectors, particularly in Section 1 and the onsite Section from the entry Guardhouse to the plant area (which are both entirely new).

Generally, there is a horizontal alignment characterized by horizontal curves with radii greater than or equal to 15m. The vertical alignment is characterized by a continuous ascent from a height of 4200 m.a.s.l. to the 5100 m.a.s.l. level at the entrance to the Mine site. Due to the existence of very sinuous and rugged terrain, a speed limit of 30 km/h has been established. In exceptionally tight curves, the vehicular speed will be reduced to speed of manoeuvring speed.

SECTION	START	END	DESCRIPTION
1	0+000	8+983	Redesigned by GMI
2	0+000	2+875	Design by ANDDES [12]
3	0+000	22+548	Design by ANDDES [1]
C.I.-1	0+000	11+925	New Section design by GMI
TOTAL		46+331	

A network of roads has been designed to connect the facilities within the mine and plant site (Figure 18-4). The final section of the mine access road connects the residential camp to the mine entrances, as presented in Section 18.1.1. (Figure 18-1). The mine entrance is located at approximately 5,000 masl on a ridge overlooking the mine and processing plant site. During construction and the first several years of mining, traffic from the mine entrance to the plant site will be on a road that is routed west around the Este Pit location, past the primary crusher, and down to the plant, which is at an elevation of approximately 4,850 m.a.s.l. (Figure 18-3).In Year 7, a new road will be established east of the Este Pit to connect the mine entrance and administrative complex to the plant site. Additional roads for access to the various components of the Process Plant, the Plant Water Pond, and other project facilities have been laid out and included in the initial capital cost (Figure 18-4).

[12] Road designed by ANDDES and reviewed by GMI

SEDGMAN

Figure 18-1: General Site Plan

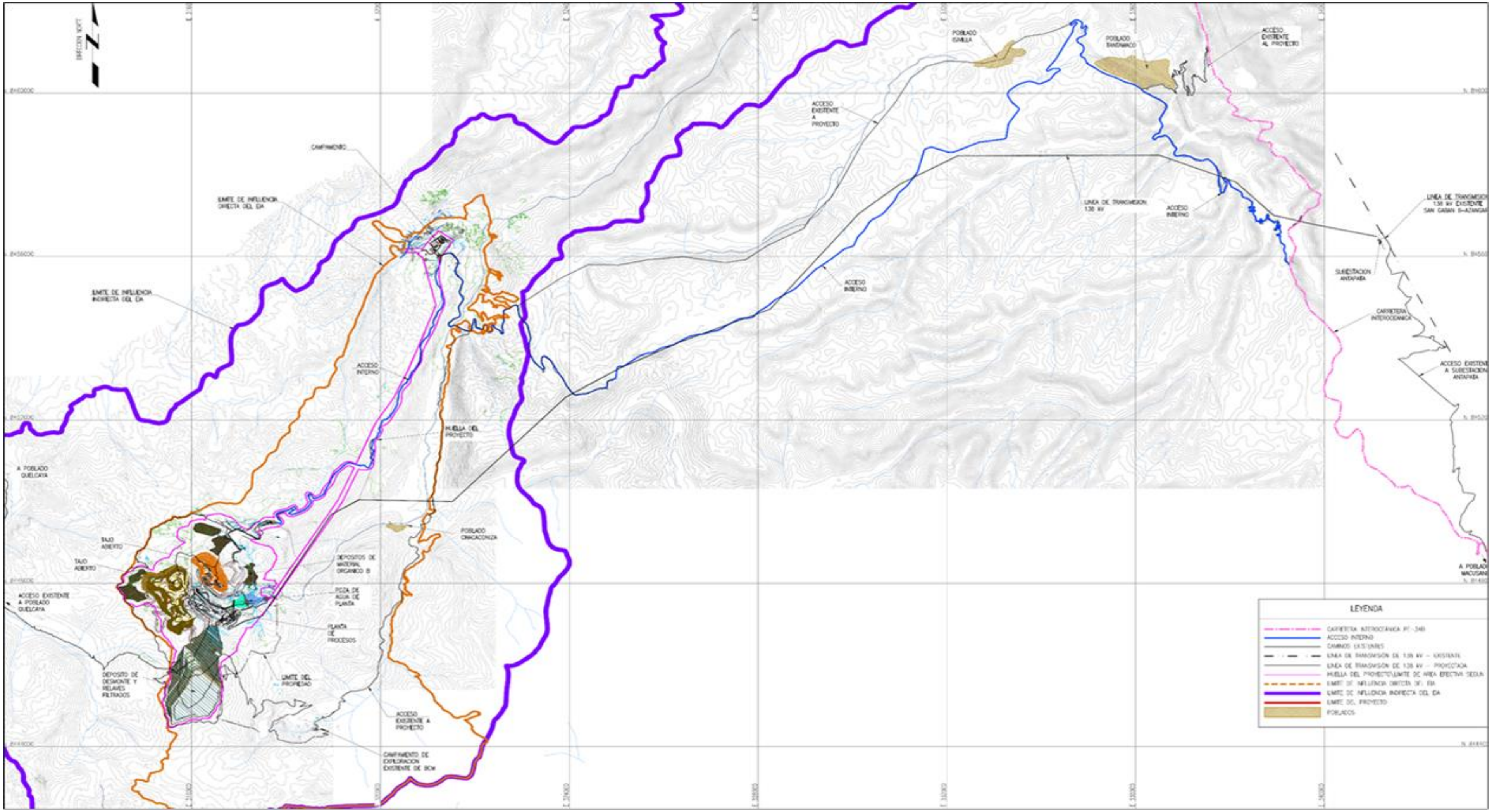


Figure 18-2: Access Road Segments 1, 2, and 3 (after Anddes, 2017)

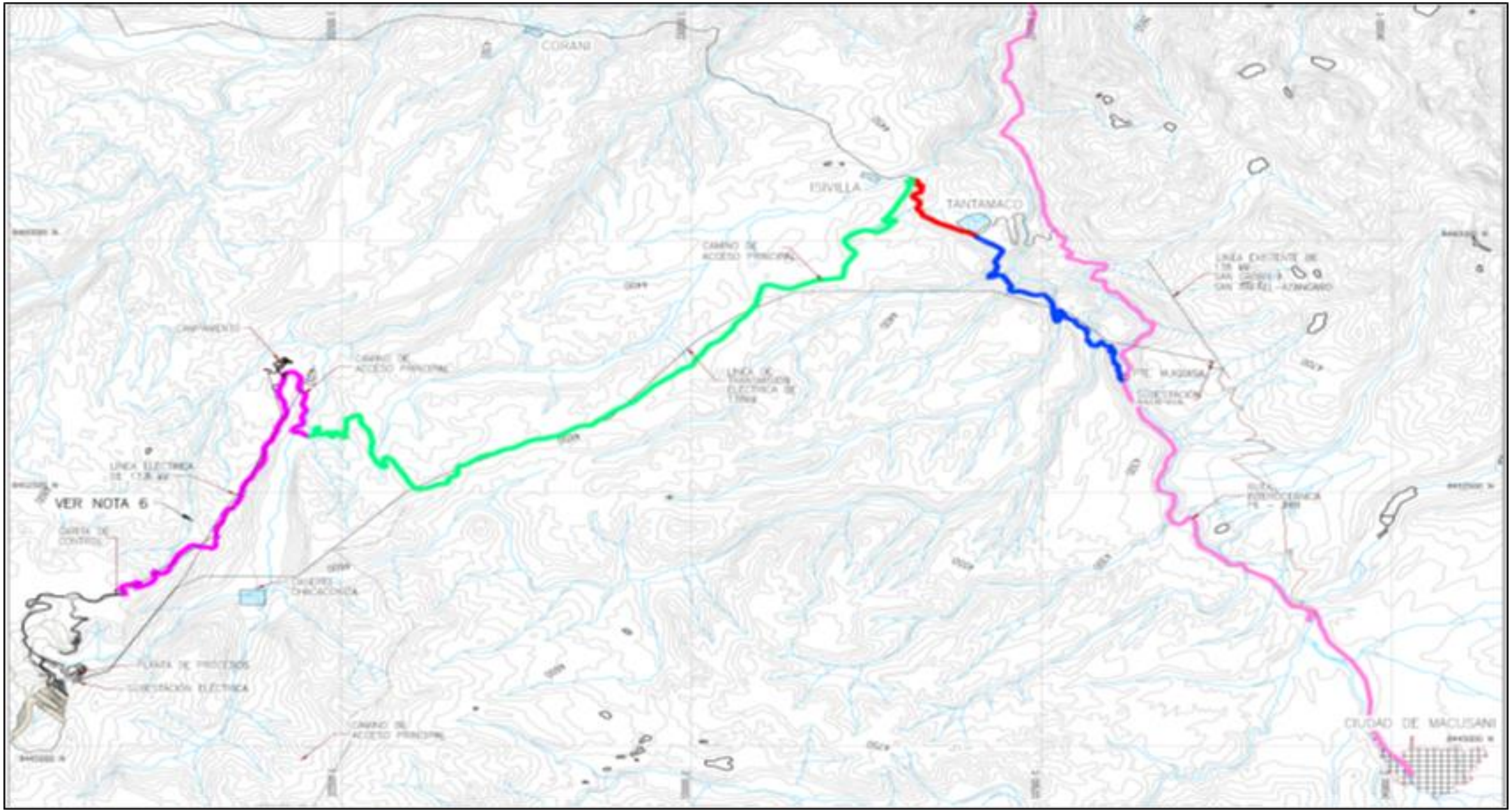


Figure 18-3: Overall Mine Site Plan

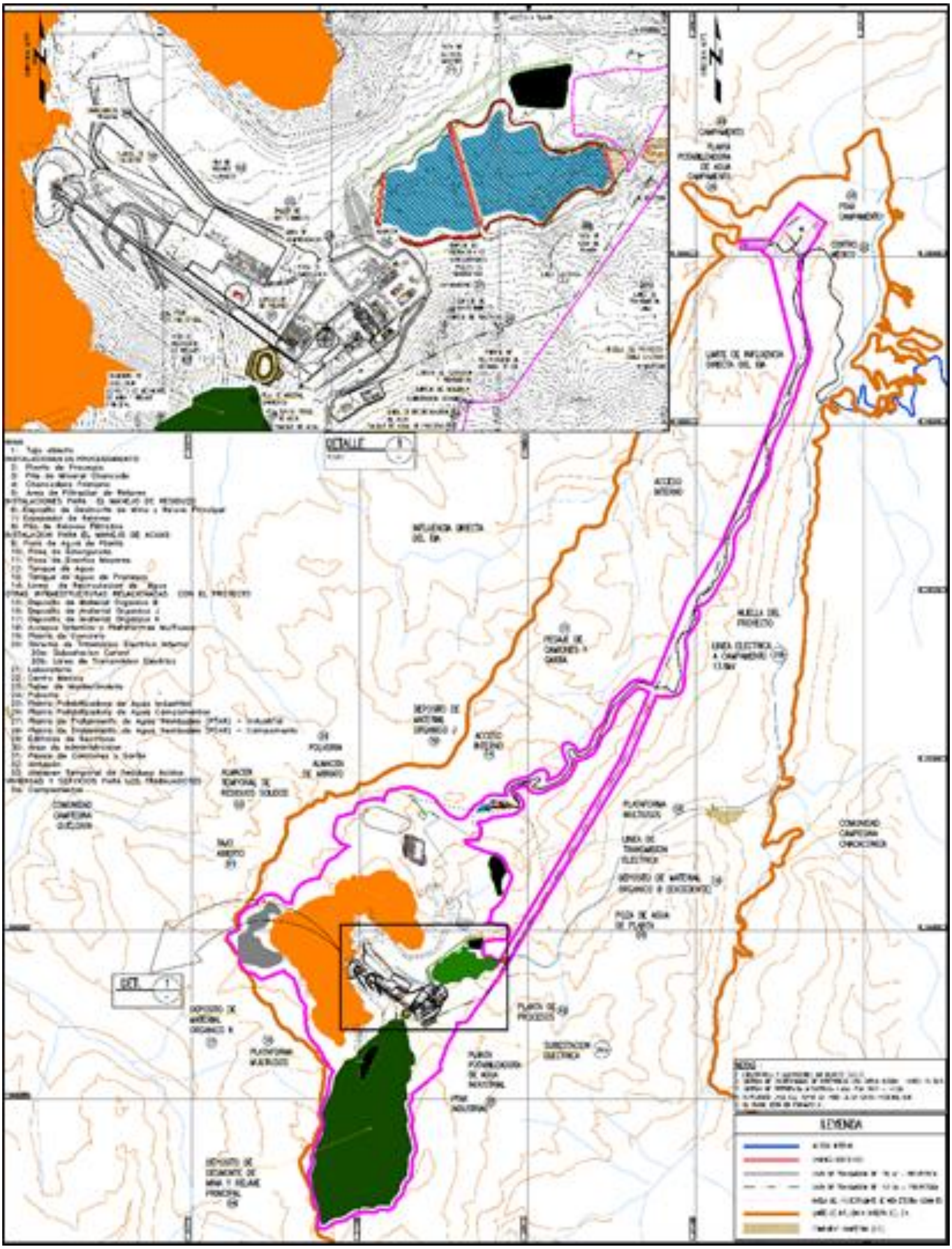
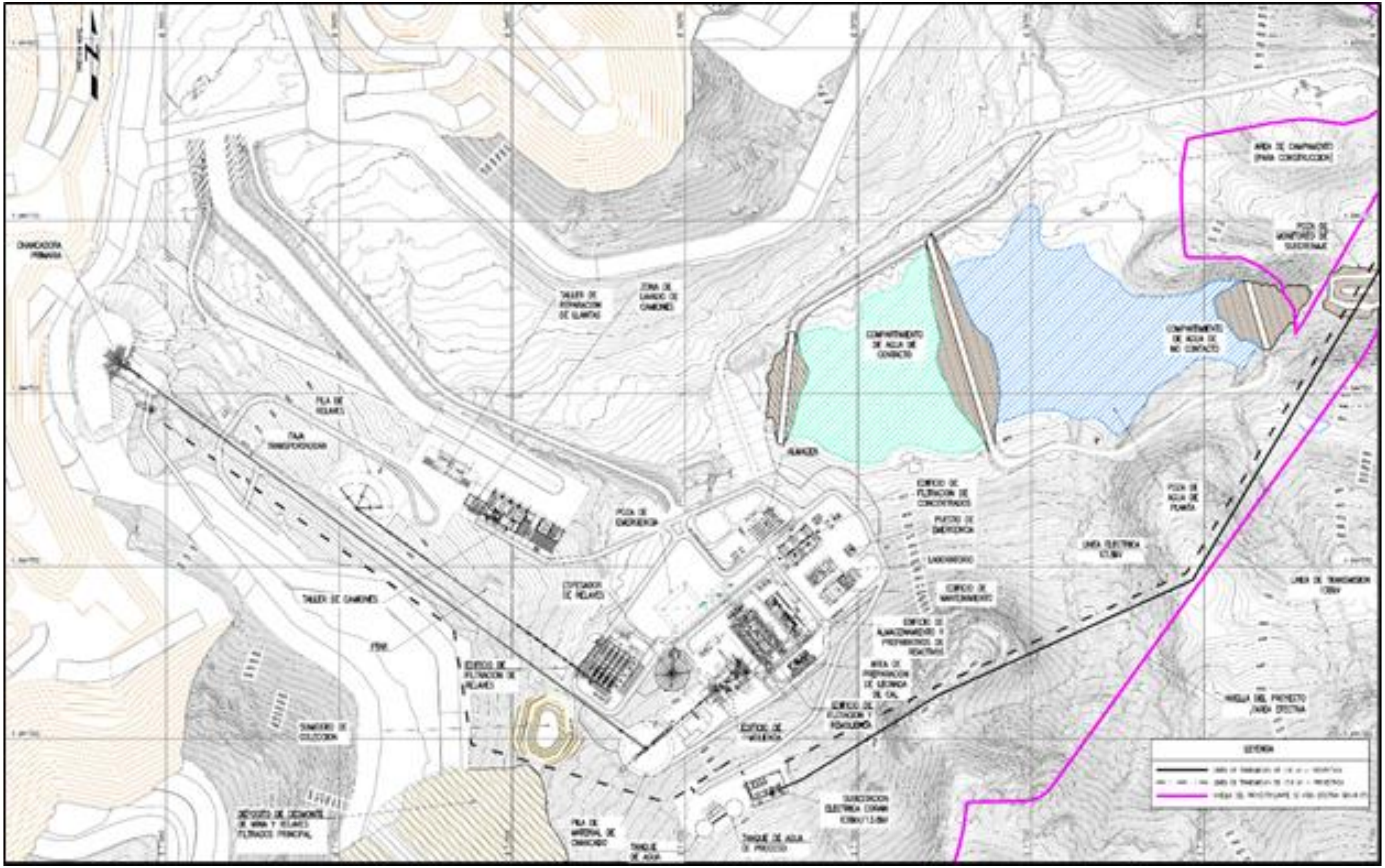


Figure 18-4: Mine Site Facilities Drawing Site Roads



18.2 Site Buildings and Facilities

Corani Project buildings and facilities are divided into four functional areas: Administration, Mine Facilities, Process Facilities, and Residential Facilities.

18.2.1 Mine Service Facilities

- Truck and Facility Workshop

  The truck workshop will be located in the north-western part of the industrial area and will have access from the mine via the primary crusher, and will also have access for light vehicles from the process plant.

  The Truck Workshop is a building, approximately 5000 m^2, intended for the preventive maintenance of heavy and light vehicles. It has 3 bays for heavy trucks and 1 bay for special equipment.

  The Truck Wash will be located next to the Truck Workshop building, and has 1 bay for heavy trucks and 1 bay for special equipment.

  Both buildings will be structural steel with metal cladding. The height of the building will depend on the final mine vehicle selection.

  The Truck Workshop will house the following:

  - Reception.
  - Dining room.
  - Heavy Equipment Spare Parts Storage.
  - Welding Equipment.
  - Electric equipment.
  - Mechanical Equipment.
  - Light Equipment Maintenance Workshop.
  - Light vehicle parts storage
  - (2) Truck washing bays.
  - (3) Maintenance bay Eq. Heavy Equipment
  - (1) Maintenance Bay Special Equipment.
  - Lubricants and Additives. Warehouse
  - Reception, 2nd level.
  - Electrical Room.
  - IT Room.
  - Training room.
  - Janitors room
  - Washrooms Men and Women.
  - Printing area 1.
  - Dispatch Room.
  - Kitchenette 1.

- Meeting Room 1.
- Individual Offices (10).
- Area for work modules.
- Meeting Room 2.
- Kitchenette 2.
- Printing area 2.
- Janitors Room 2.
- Men’s Washroom and Changing Rooms.
- Women’s Washroom and Changing Rooms.

- Fleet Management System

  A fleet management system has been included. The system considered is a system developed in Peru, and has been previously implemented in some mining operations in that country. The Supplier will provide all necessary hardware and software and operate the system for the first five years from the start-up of operations with mining equipment. From year six, the system will be operated by the mine operations personnel of the Bear Creek Mining.

18.2.2 Administration Facilities

The administration facilities are located near the main entrance partially because of space constraints and partially to keep suppliers and non-essential personnel out of the mining and process areas. The administration facilities include the main gate and guard house, an administration office building, and a warehouse to receive parts and supplies necessary for operation and maintenance.

- Guard house and weighbridge

  The main Guardhouse and Plant entry at the entrance of the industrial area, will be composed of a booth for the control of personnel, as well as an area for the control and registration of vehicles. The guard house will contain security offices, restrooms, and a small reception area. A Peruvian security firm will be contracted to provide on-site security services to the Project starting at the time of preproduction. The area also includes a weighbridge (Scale), to weight trucks of supplies and concentrate.

- Fuel supply

  For the life of the mine, a fuel station will be built near the Main Access Gate. This fuel station will have 378m^3 storage capacity (equivalent to 9 days of storage during peak demand in 2021) and will be equipped with two fuel pumps for light equipment. A contracted fuel supply company will be responsible for fuel supply and lubrication of all mining equipment in the field, using two hybrid trucks (fuel / lubrication) that will operate 24/7. Equipment from other areas will also be supplied at this fuel station

- Warehouse Building

  The warehouse building will be a pre-engineered structure of about 900 m^2 with metal roofing and siding, supported on a metal structure. Outside will have the truck manoeuvring area and the warehouse yard used to store materials that are not affected by climatic conditions. The warehouse is on a raised levelled area at 1.20m above the natural terrain, in order to facilitate the unloading of trucks.

  The building has different accesses for plant personnel and deliveries. Inside the main building there will be a separate area enclosed by wire mesh for small high value small articles.

  - Within Main Building (roofed area), main storage area
  - Small item storage

  - Offices
  - Washrooms
  - Reception.

  Outside areas include:

  - Materials receiving area including truck reception platform, manoeuvring area and storage yard

- Administration Building

  The Administration Building is located close to main entrance and access road and includes offices, a reception area, a conference room, a training room, and a first-aid room. The offices will be used for senior personnel, records and archives, accounting, and engineering. Other areas of the building contain conference rooms, restrooms, and a reception area. This facility will be a heated/air conditioned pre-engineered metal building with insulated roofing and siding, and installed on a concrete mat foundation. A gravel surfaced visitors and employee parking area is also provided.

18.2.3 Process Facilities

The process plant facilities include a primary crushing facility located close to the mine, crushed ore stockpile area located close to the concentrator area, a grinding building, a flotation area, a concentrate dewatering and load-out facility, a tailing storage facility, a plant maintenance building, administration building, and laboratory building.

- Primary Crusher

  The primary Gyratory crusher structure will be of reinforced concrete and structural steel construction. The dump pocket is designed to receive two CAT 785 haul trucks at the same time. A hydraulic rock breaker for reducing oversize mined material is provided. Crusher maintenance will be performed with mobile cranes.

- Crushed Ore Storage and Reclaim

  From the primary crusher, overland conveyors will transport the crushed ore to a crushed ore storage facility which will be an open stockpile with no superstructure. A concrete tunnel will be provided beneath the stockpile to house reclaim feeders. The feeders discharge onto the SAG mill feed conveyor that connects the stockpile to the mill building. A dust extraction system will be provided to control the emission of fugitive dust. All transfer points between belt feeders and the SAG mill feed conveyor will be connected to the dust extraction systems through ducting. Belt conveyors outside the tunnel and building will have belt covers to minimize dust emissions.

- Grinding Area

  The Grinding Building will house a SAG grinding mill, ball mill, primary cyclones, mill discharge screen and primary cyclone feed pumps in an engineered structure with metal roofing and siding. Internal structural steel platforms that provide maintenance access are supported independent of the main building structure. The building foundations will be concrete spread footing, piers and grade beams. Concrete floor slabs will be provided over the entire area of the building with curbed containment for spillage control and thicknesses suitable for maintenance traffic where appropriate. Overhead travelling cranes are provided for maintenance of the grinding equipment. Controls rooms and offices within the structure will be air conditioned/heated, with high-bay lighting provided throughout.

- Flotation Area

  The flotation circuit has an engineered structure with metal roofing and siding. This building will house four phases of (air-induced) flotation; pump systems, lead and zinc regrind ball mills, and zinc, and lead concentrate thickeners.

Concrete floor slabs with curbed containment are provided. Flotation tanks will be elevated on concrete and structural steel supports and cell covers and structural steel access platforms will be included. Maintenance on the flotation circuit will be facilitated by two cranes to serve the lead and zinc sections separately. High-bay area lighting will be provided to illuminate the area.

- Concentrate Dewatering and Loadout

  Concentrate filtering and load-out facilities will be located within an engineered structure with metal roofing and siding. The filter area includes two Larox-type filters: one filter for lead, and one for zinc. Filtered concentrates will be stored in segregated piles underneath the filters. Internal structural steel platforms that provide maintenance access are supported independent of the main building structure. The building foundations will be concrete spread footing, piers and grade beams. Concrete floor slabs will be provided over the entire area of the building with curbed containment for spillage control and thicknesses suitable for the maintenance and concentrate transport truck traffic where appropriate.

  Filtered concentrates from the stockpiles will be transferred to bagging machines to prepare the concentrates for loadout and shipment to market. The load-out facility has a covered area and loading docks to load highway transport trucks with a fork lift.

- Tailing Filtering Facility

  The tailing filtering facility includes a thickener, agitated filter feed tanks, and recessed plate filters for dewatering the tailings. Support equipment includes compressors to blow air through the compressed filter cake at the end of the cycle and conveyors with a radial stacker to transport the filtered product to a stockpile area. Trucks will be used to haul the filtered product to the Main Dump area for co-deposition with waste rock from mining operations.

- Reagent Area

  A pre-engineered reagent mixing building with metal roofing and siding will be provided to store, prepare and supply reagents to the process. The reagent mixing and storage facility will be located adjacent to the flotation area to receive and store reagent supplies. The reagent building includes a truck unloading area, inside storage, mixing tanks, distribution tanks, an overhead crane, and concrete with containment and floor sumps. Cyanide is kept in a separate area in accordance with Peruvian regulations.

- Plant Maintenance Building

  The Plant Maintenance Building will be a pre-engineered structure with insulated roofing, siding, and a reinforced concrete mat foundation. The building is located within the plant area, and contains the main plant administrative offices (2nd level), and a lunchroom and plant personnel change rooms on a first level. The building area is about 1238 m^2.

  The administrative section of the building has two levels, which permits the ground floor maintenance area to be of a greater height. The first level contains public access areas such as the dining room and change rooms. The rest of the offices are on the second level.

  The maintenance area has access for vehicles, as well as storage for supplies and tools.

The Workshop Building/Plant Offices includes the following:

First level

- Dining room (Cap. 30 / shift).
- Washroom and Women's change room.
- Washroom and Men's change room.
- Lockers
- Reception.

- Waiting area
- Maintenance workshop.
- Electrical and Instrumentation shop
- Tool Storage.
- Electrical room.
- Machine room.
- Compressor room.
- Welding area.
- Wardrobe area.

On the second level are the following:

- Private offices and workstations.
- Meeting room.
- Kitchenette.
- Board / Data Rooms.
- Printing area.
- Washrooms, Men / Women

- Analytical Laboratory

  The laboratory building will be a pre-engineered structure with insulated roofing and siding, and a reinforced concrete mat foundation. Laboratory facilities include sample receiving, sample drying, sample preparation, metallurgical lab, wet lab, fire assay, electrical and mechanical rooms, a computer room, men's and women's restrooms and locker facilities, a lunch room, a loading dock, and various offices. The building will be heated/air conditioned, and fume extraction and dust collection equipment is provided.

- Fuel Stations

  In addition to the mine fuelling station located in the Administration Area, two other fuelling stations will be provided. The first is a temporary construction installation, with modular re-fuelling points and a 76 m^3 storage capacity, used during earth moving operations until the permanent mine fuelling station is constructed.

  The second is in the camp area, as a supply point for project personnel transportation vehicles, it will have a 113.5 m^3 capacity.

- Hospital

  The hospital is intended for primary medical care for project personnel, as well as occupational health. The building is located in the camp area from the early stages of construction. For that reason, a prefabricated modular construction system has been proposed for quick installation.

  The Construction Manager, through a service contract with a company specialized in health services, will provide facilities, personnel and equipment to support and attend to emergencies during construction. This service will include the incorporation of (1) Type II ambulance unit, equipped and authorized by the holder of Department of Health (Technical Standard of Health for the assisted transport of patients by Land, approved by Ministerial Resolution No. 953-2006-MINSA), for the transport of patients to the hospital of Macusani (Hospital of Level II-1 PIP 129661).

The constructed area of the hospital will be 185 m^2, including the following indoor environments:

- Triage, Reception, Topical.
- Offices (2) + Pharmacy.
- Observation and Trauma.
- Bathroom, Observation and Trauma.
- Washrooms
- Occupational Health and Security.
- Occupational Health and SH.
- X-ray.
- Occupational Health - Files.

- First Aid Post

  The First Aid Post is located inside the Operations Area to attend to emergencies and for the preventive activities of the First Aid and Rescue Brigade It contains the brigade offices as well as parking for rescue vehicles. The building area is approximately 205 m^2.

18.2.4 Residential Facilities

The distance of the Project site from any significant urban area that could provide lodging and services to mine personnel requires that the Project include a residential camp. The camp site is located at an elevation of approximately 4,400 m.a.s.l., about 10 km to the northeast of the mine facilities. The UTM, Zone 19S, Datum PSAD56 coordinates of the site are: 8,456,510 m North; 321,312 m East.

Figure 18-5: Camp Location

The camp has been organized in such a way that the facilities are located in a central core. The core/circle design is to offer higher protection against cold wind and other inclement weather.

Parking areas are centrally located close to the dormitories. Vehicle traffic is limited in dormitory areas, and only design for pedestrians is considered. The inner circle distributes traffic for facilities centrally located for delivery of supplies.

The external circle is for ease of pick-up of workers to transport them to the mine and return to the camp.

Design is based on the construction camp being converted to the operations camp the project considers being able to have a construction camp in the first years, so that it can be used in the operation phase and it will be built according to this concept of reuse, in order to optimize the infrastructure to be included.

Camp capacity was estimated based on the projected occupancy curve resulting from the construction program and the estimation of permanent staff in the operation stage.

Figure 18-6: Camp Layout

The following construction systems are considered according to the concept proposed:

Permanent Buildings: Composed of mixed prefabricated systems (concrete slab with insulated panelling on metal structures), as well as modular prefabricated systems. These are the buildings that will be used in the construction stage and later re- used during operations.

Temporary Building: Prefabricated modular buildings and tents. These are the buildings that will be used only in the construction stage due to their quick installation, relocation and quick removal at the end of their use. The facilities are designed to accommodate 2,180 persons during construction, falling to 652 persons during operations

The operations camp includes the following buildings:

- Cafeteria and kitchen;
- Laundry;
- Recreation facility;
- Store;
- Medical facility;
- Camp administration;

- Camp sewage treatment;
- Camp water treatment;
- Camp maintenance;
- Dormitory Type 1, 12 men;
- Dormitory Type 2, 24 men
- Dormitory Type 3, 64 men
- Dormitory Type 4, 36 men
- Camp guard/security house; and
- Recreation area (open area).

The Camp will have the following facilities and services:

- Distribution and storage system for drinking water, including water collection.
- Drainage network including modular treatment plants according to the camp requirements. Separate systems and treatment have been included for the laundry and kitchen.
- Exterior fire water network.
- Hot water system in buildings (electric heaters) for accommodation, laundry, kitchen and health hospital
- Gas supply for kitchen.
- Exterior lighting systems.
- Atmospheric protection system.
- Grounding system.
- Emergency electrical system (Generator for hospital and refrigerated refrigerator and freezer for the kitchen).
- Fire extinguishers in all buildings.
- Cable TV system in accommodation, recreation and dining room.

18.2.5 Communications

- Radio Communication System

  The analog radio communication system will operate in the VHF frequency band. It will have the ability to communicate with radios located in all areas of the Mine, Plant and Administration Areas, across the project footprint. It will include a base station and the repeater stations that are necessary for its operation, and the provider will procure all permits and licenses for the use of the frequencies that are needed.

  At least 5 radio channels (surveying, operations, supervision, maintenance and emergency) will be required for the Project

- Voice and Data Communication System

  The voice and data communication system will be installed in the different administrative offices, hospital, dining rooms, recreation, accommodation, control rooms and checkpoints, All buildings will include conduits and trays for cables (voice and data) and for the power supply for the telecommunications cabinets.

  The telecommunications cabinets will be distributed in different areas of the plant. The cabinets will contain the voice and data network, equipment and accessories, the network of the fire detection system and the controls network. Transmission is via interconnecting fibre optic cables.

- Cable TV Distribution System (CATV)

  Buildings will include conduit required for CATV system. These will be independent of the conduit for other systems, but they will be able to share cable tray, provided metal separators are installed.

  The CATV system supplier will supply all the equipment, licenses and accessories for the reception and distribution of the TV signal in the housing modules of the camp.

  The CATV channels to be transmitted shall be submitted by the service provider for evaluation of CORANI

- Storm Detection System

  A Storm Detection System will be installed that is capable of detecting storms and generating at least 3 alarms (15 Km, 10-5 Km and less than 5 Km distance).

  Two detection sensors will be included, one in located in the camp and another in the plant. Each sensor must have the capacity of detecting a storm at 20 km distance as a minimum. Each sensor will be connected to a control panel from which alarms will be sent to strobe lights, sirens and to an operating station located in the camp security gates.

  Red strobes will be installed, which will activate when the storm is less than 5km away. In the camp, a minimum of 10 strobe lights will be connected to the storm sensor controller. For the process plant, 06 strobes with siren will be required.

  All alarms and sensors will be powered by 120 VAC supplied from the instrumentation panels located in the electric rooms.

  The alarms will be located at front of the doors of the buildings, in pedestrian zones and in parking lots.

18.2.6 Water Supply and Management

Surface water and groundwater will be used to supply the water needs for the Corani Project. Water is classified in two ways: either as contact water or non-contact water. Contact water is water that comes in contact with mine operations or rocks capable of generating ARD. Contact water includes pit dewatering water, runoff from the open pit, and seepage from the main dump. Non-Contact water is generated from runoff from unimpacted ground surfaces. Both contact and non-contact water are stored until required in a dual-cell water pond located near the plant. The contact water cell can hold 257,000 m^3, and the non-contract water cell can hold 572,000 m^3.

Section 20 describes the site-wide water balance used to verify that the plant has sufficient water in extreme dry conditions. The plant water requirements are drawn from either the contact or non-contract water pond depending on the water quality requirements of the plant. The issues of surface water management, water supply, fire protection, sanitary waste management, and are presented in the following sections.

- Surface Water Management Plan

  The site surface water management plan was formulated to provide a template for management of storm water runoff and stream flows in the Project area. The effective management of surface water resources at the Project site is critical to the protection of water resources downstream of the Project area. In general, surface water will be managed to separate contact water and non-contact water. Contact water is stored and consumed. Non-contact water will generally be diverted around the project facilities, except when mineral processing needs dictate the use of some or all the non-contact water.

From the Site Water Balance (see Section 20), the sources of Contact Water are:

- Pit dewatering water.
- Drainage from mine haul roads.
- Water due to precipitation in process plant area and other facilities.
- Precipitation over the Contact Water compartment in the water pond.
- Seepage from the Main Dump (this is primarily water extracted from the consolidation and compaction of mine tailings).
- Water collected in the organic material storage, the surplus material deposit, and the solid waste storage areas.

Non-Contact Water sources are:

- Water from creeks and valleys that have not had any contact with areas disturbed by the Project.
- Precipitation above the non-Contact Water Reservoir in the water Pond.
- Water runoff from precipitation in all undisturbed areas of the Project.

The consumption and losses in the Water Balance are as follows:

- Evaporation,
- Water replenishment for the Process Plant: 201.1 m^3 / h, according to the Process Plant Water Balance.
- Water for suppression and dust control on roads and in the mine
- Permanent discharge of Non-Contact Water to maintain flow in the Chacaconiza Creek 2.6 L / sec.

Major surface water conveyance and storage structures have been designed using hydrology and hydraulics predictive models. Data used in the models was derived from previous and ongoing site investigations. Conservative assumptions have been applied where possible.

Development of the mine pits will be completed sequentially, with site conditions changing frequently. To effectively manage evolving site conditions, surface water management features and designs have been presented in "snapshots" for various years during the mine-life, including the preproduction and closure periods. In practice, the surface water management features will be re-evaluated and expanded as necessary each year during the dry season in order to accommodate the expanding project footprint.

During construction, contact water will be directed to settling ponds for recycling and consumption (as dust suppression water), as necessary.

At closure, contact water will be minimized by reclamation activities. However, portions of the mineral processing plant will be converted to a water treatment plant to treat any ARD impacted leachate or runoff.

Diversion ditches, culvert systems, ponds, sumps and pipelines, have been designed to address the majority of surface water flow at the project site. In addition, best management practices (BMPs) will be employed to minimize erosion and sediment transport as well as deposition of sediment at the project site. Erosion control BMPs will include the use of temporary diversion ditches, check dams, rock-containment berms, straw wattles/coir logs, silt fencing, terraced and bermed slopes, ditch linings, riprap, erosion matting/blankets, and rock or geotextile covers.

- Contact and Non-Contact Water Pond Design

As mentioned above, the Non-Contact and Contact water pond are located northeast of the plant complex (Figure 18-5). The ponds have approximately 782,000 m^3 of storage, equivalent to approximately 5 months of makeup demand.

The Non-Contact Water pond and Contact Water Pond are adjacent to one another (Figure 18-5). The Contact Water Pond is upstream and can store approximately 225,000 m^3 of contact water. The Freshwater Pond is downstream and is capable of storing approximately 557,000 m^3 of non-contact water. Zoned embankment dams will partition the basin into two ponds to manage contact and non-contact water separately. The two-pond system is intended to minimize the amount of contact water produced at the project while still ensuring a dependable water source is available during the dry season. The dams will have a low-permeability core, compacted rockfill shells, a seepage cut-off, and internal drainage and filter control. The dam separating the Contact Water Pond and Freshwater Pond will be 15 m high, with a 1 m freeboard allowance. The Non-Contact Water Pond embankment dam will be 35 m high, with a 2m freeboard allowance. A spillway will allow non-contact water to be released downstream. Water will be pumped from the contact water pond to the plant for use in the process circuit.

The pond size has been determined by the Site Wide Water balance (see Section 20). It is designed from the results of a probabilistic water balance in accordance with best-industry practice. It accommodates enough water to contain (for consumption) the 95 percentile wet conditions, and enough water to supply the mine in the 5% dry conditions.

Additional geotechnical work is required to build the dams (see Section 26).

The pond system will be constructed prior to other facilities to serve as a construction-phase sediment pond. The system will include a low-level outlet to release water during the construction phase, and a spillway to release water during operations.

Site-Wide Water Conveyance and Management

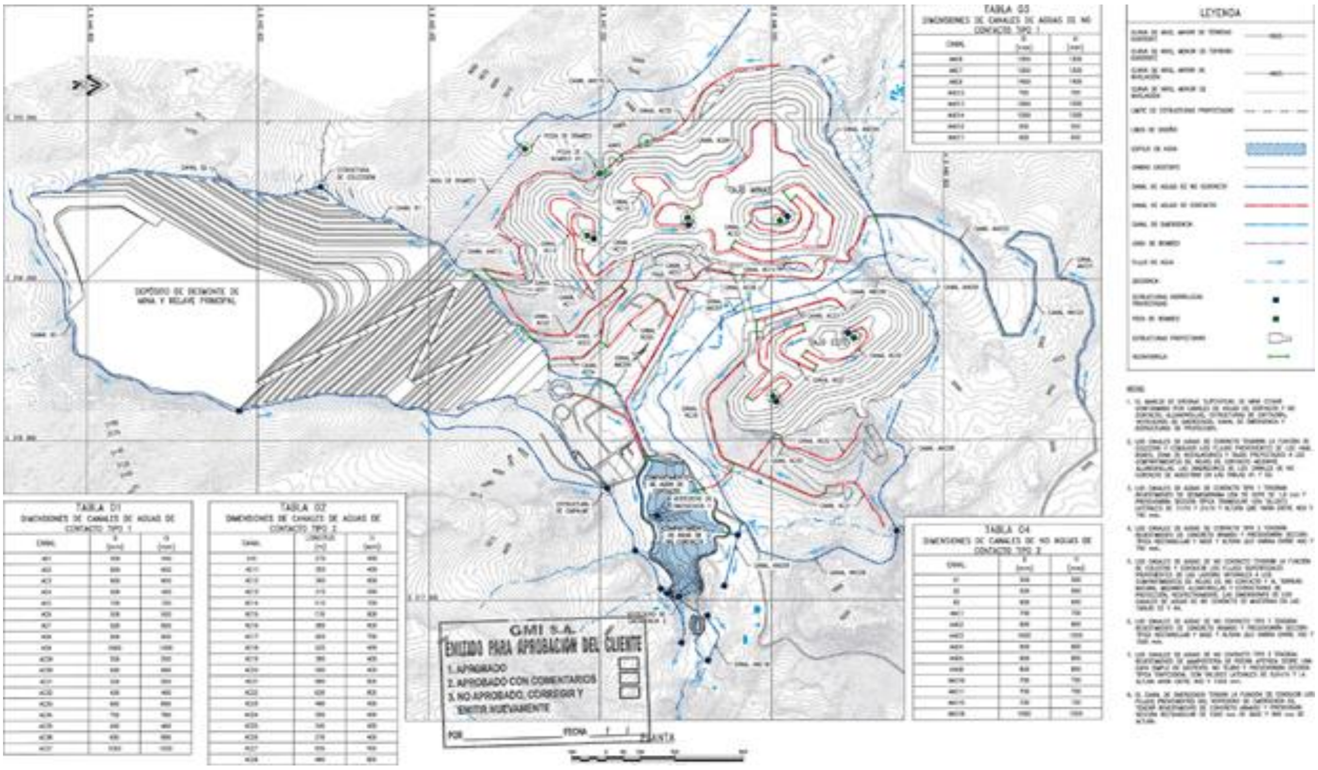
Precipitation runoff internal to the plant facilities is a minor percentage of the total water supply requirement. Runoff in plant areas will be collected in small sumps and recycled to the Process Water Tank.

Pit water will be collected in sumps excavated into the pit floor. Sumps are configured as a simple box-cut into the un-mined bench below the active pit floor. Water will be pumped from the sumps to the plant via skid-mounted pumps and high-density polyethylene (HDPE) pipelines.

Seepage and runoff from the main dump will report to . a sump will be located at the toe of the Main Dump, and water will be pumped from there to the plant or the contact water pond via skid-mounted pumps and HDPE pipelines.

Water may be released from the non-contact water pond during operations when necessary. The pond system includes a bypass to allow stream flows in excess of water supply demands to be routed around the pond and released to the natural drainage downstream.

Figure 18-7: Contact Water Pond and Freshwater Pond



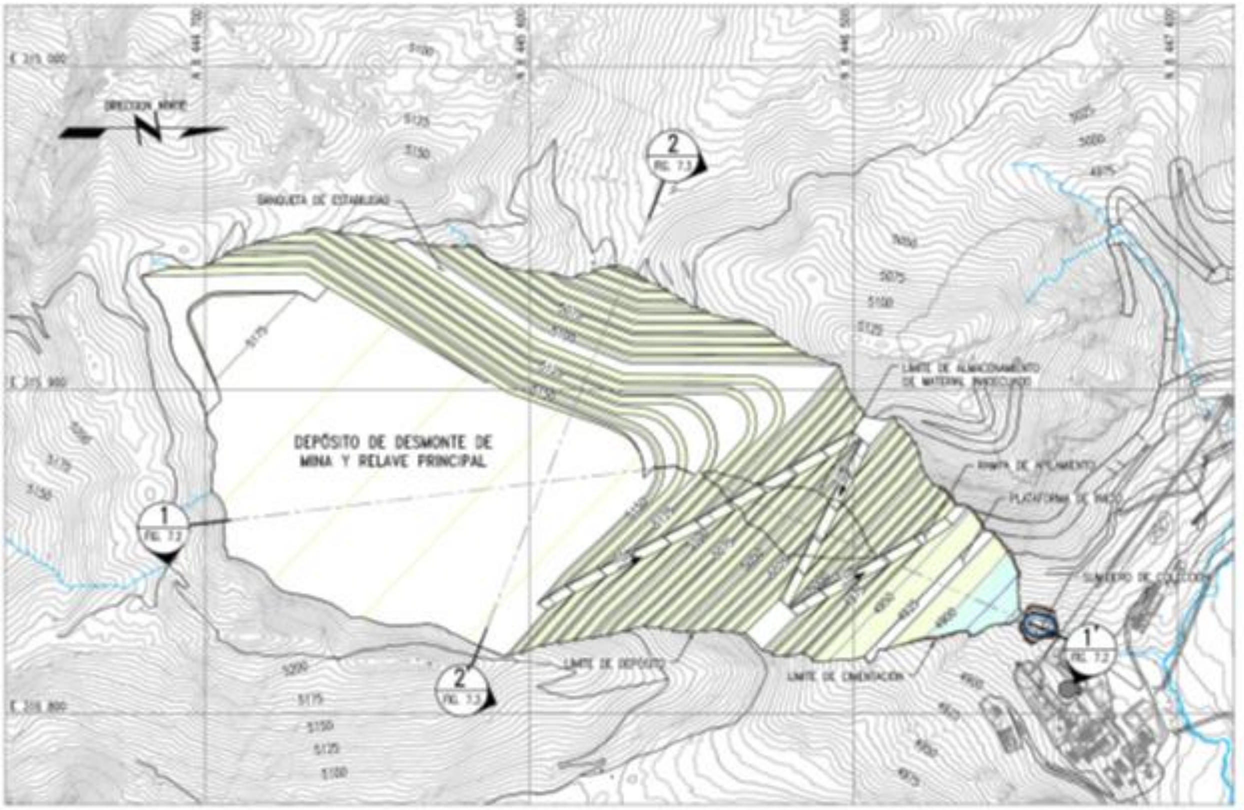
The project will utilize a Fresh/Fire Water Tank at an elevation of approximately 4,900 m.a.s.l.. Water for this tank will be drawn from the non-contact pond. Fresh water for plant use would be drawn from the upper nozzle on the tank to ensure an adequate reserve for a fire water supply. Fresh water will also be drawn from the upper nozzle for the drinking water treatment plant to supply potable water to sinks, eye-wash stations, and drinking fountains throughout the facilities. A nozzle at the bottom of the tank provides water for fire suppression (Section 18.3.2).

Fresh water for the Administration area and camp area (near the main entrance) will be taken directly from the Imagina Mayu River at coordinates 320,600 East, 8,456,155 North, through a catchment and pumping system located on the shore. The water will then be pumped to a Fresh Water Tank located in the Camp. The fresh water will be treated for sanitary use and cleaning. Additional treatment will be provided for the kitchen. The water for human consumption will be provided from the fresh water tank at the 5110 elevation described in the paragraph below.

The Fresh Water Tank is at 5,110 m.a.s.l. Water from this tank will provide freshwater and fire suppression for the Administration area. Water for camp use will be drawn from the upper nozzle on the storage tank, while water for fire suppression will be withdrawn from the bottom nozzle to maintain sufficient capacity in the lower part for fire suppression at the camp.

All wastewater from the Camp will be collected through a drainage system and taken to a Wastewater Treatment Plant (WWTP) also located inside the camp. Wastewater will be treated to comply with the corresponding LMPs to be discharged to the Chacoconiza river at the coordinates 322,079 East, 8,456,150 North.

Figure 18-8: Contact and Non-Contact Water Collection Network



- Fire Protection

  A fresh water pipeline will deliver up to 40 m^3/hr of fresh water from the Plant Water Pond to the project Fresh/Fire Water Tank, located at the project site at an elevation of approximately 4,875 m.a.s.l. The Primary Crusher and mine services area (Truck Shop/Fuel Facility) are areas are too low in elevation to receive fire suppression water by gravity feed. As a result, a fire pump will be required to provide water at sufficient volume and pressure for fire protection for these facilities.

  Fresh water for plant use would be drawn from the upper of two nozzles located on the tank at an elevation high enough to ensure an adequate reserve for a fire water supply. From the lower nozzle, a buried firewater loop will supply fire water to hydrants located throughout the concentrator area, the ancillary buildings, and the laboratory.

  Individual hand-held fire extinguishers will also be located throughout the offices and work areas. In addition, a fire truck will be available to service the camp site as well as the main site.

  The off-site Administration Building, Gatehouse and Security Office, located close to the Mine Access Road and entrance to the mine, will be provided with hand-held fire extinguishers. These extinguishers will be distributed throughout the offices and common areas in accordance with North American and Peruvian Fire Codes.

- Sanitary Sewage

  A packaged, activated-sludge wastewater treatment plant will treat sewage from the process facilities, laboratory, warehouse, plant maintenance building, truck shop, and tire shop. Treated effluent goes to a septic field next to the wastewater treatment plant.

  The camp facilities, the administration building, and the security office and gatehouse located close to the Mine Access Road will also be served by a package, activated-sludge sewage treatment plant, and the treated water will be disposed of through an adjacent septic field.

18.2.7 Power Supply and Power Distribution

A new 138 kV power transmission line is necessary to provide power to the Corani project. A new power substation will connect with Power Transmission Line L-1013 (San Gabán II – San Rafael – Azángaro) as the power source. A new 138 kV power transmission line will be built to connect the Antapata substation to the Main Corani substation to be built near the Project's main process buildings. The proposed alignment for the 138kV line (Figure 18-1) was provided by Promotora (2015). The transmission line route was selected based on using the route already provided by the Project's Mine Access Road.

Power will be distributed from the Main Corani Substation from a pair of step down transformers (one in use, one, standby) to the distribution voltage of 13.8 kV. Both overhead and underground power lines will be used to distribute power throughout the plant site and administration area. A 13.8 kV transmission line with a length of approximately 13 km will connect the mine site to the residential camp site.

- New Antapata Substation

  This substation will be connected to the San Gabán – Azángaro 138 kV Transmission Line at approximately 55 Km from the San Gabán Substation. The new Macusani substation will include the installation of three (3) switch yards at 138 kV.

- Transmission Line

  The link between the Macusani and Corani substations is made through a 138 kV – 60 Hz transmission line, approximately 37 km in length. The line includes:

  - A metallic self-supporting lattice towers;
  - Aluminum alloy conductors (AAAC) with a 300 mm^2 nominal section;
  - Two ground wires (one steel galvanized wire and another OPGW-type wire with 16 optical fibers);
  - Glass insulators with a 146mm spacing and a 280mm diameter;
  - 380 mm leakage distance; and
  - A 160 kN electromechanical rupture load.

  The projected transmission line has the following technical characteristics:

  - Rated voltage: 138 kV
  - Maximum voltage: 145 kV
  - System Frequency: 60 Hz
  - Length: 29,6 km
  - Design power: 65 MW
  - Number of circuits: 01
  - Number of conductors / phase: 01
  - Circuit layout: Triangular
  - Guard Cable: 02, Type EHS and OPGW
  - Conductors: Type AAAC
  - Structures: Self-supporting towers of A ° G °
  - Insulators: Tempered glass

  The insulator chains in the suspension are made up of fifteen (15) insulators and the insulator chains in the grounding have sixteen (16) insulators and a reinforced concrete foundation, forming a set of four separate columns. Each column will be made of corrugated steel-reinforced concrete with a formulation of f'c=210 kg/cm^2

- New Corani Substation

  This substation will require the installation of a 138/13.8 kV transformer. The 138kV side will have a power switch and the power transformer, for an installation altitude of 5000 masl, will have an automatic regulation load of:

  138 ±10x1%/13.8/4.16 kV - 48/48/16 MVA (ONAN) - 60/60//20 MVA (ONAF)

  It will also have a YNyn0(d1) connection (or vector) group as well as a regulation board. It also includes six cells at 13.8 kV and an automatic capacitive reactive compensation bank of 5 MVAR.

  Power will be distributed from the receiving substation at the Project site through underground duct banks to nearby major loads via local substations. Power distribution to all other areas, such as tailing water reclaim, the administrative services building located close to the concentrator, laboratory, fresh water pumping, crushing station and truck shop, and to the mine, will be via overhead 13.8 kV power lines.

18.2.8 Communications System

The project off-site telecommunications will be served by a fiber optic telecommunications link to a data centre in Lima that will be underbuilt on the 138 kV transmission line from the Antapata substation. Telephone and data communications including voice, data and internet communications will be provided for the mine site and Macusani. The communications system will connect to a central communications centre, which will include a telephone/fax PBX and network servers for email, internet and data services. Other network servers to manage site operations and for data storage will also be located in the central communications centre, with the exception of the process servers which will be located at the processing facility. The mine site telephone system will link all essential areas of the site together, and through the satellite system, to outside of the project site.

The mine radio system will include one base station and a control-tower station at the mine from which all mining equipment and haul trucks will be dispatched and controlled, and a number of repeater stations will be installed. One station will provide coverage to the tailing area, and others are required to extend coverage throughout the mine site and to Macusani.

All vehicles will be equipped with radios and essential personnel will have hand-held radios. Key personnel will also be equipped with mobile telephones. Cellular phones will have coverage to Macusani as a safety precaution.

18.2.9 Waste Rock and Tailings Management Facilities

Detailed Engineering of the Mine Waste Rock and Tailings Deposit has been carried out based on conventional technology for disposition of mine waste rock and process plant tailings. The basic design criteria prepared by Anddes have been used as a reference, throughout the project and are included in Chapter 2 of Document 1105.10.11-5-100-02-ITE-001

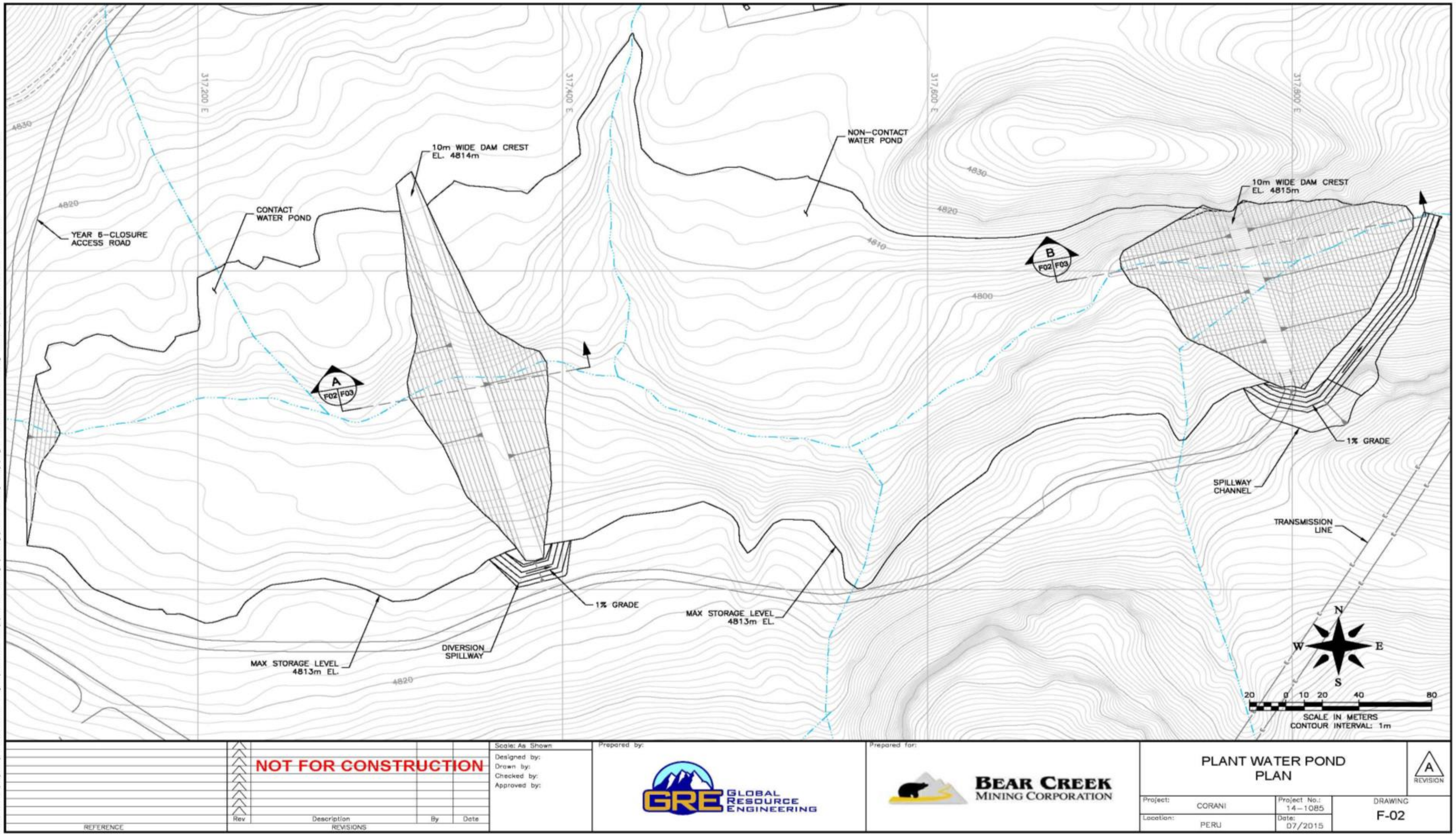
The civil design of the deposit includes the levelling of the foundation, removal of organic material, construction of the sub-drainage system and seepage, collection pond and disposal plan for mine waste rock and process tailings by year, and the dump access ramps for dumping of waste rock and filtered tailings.

The design is based on local slopes of 2,2H: 1V and global 3H: 1V, in accordance with stability analyses and to criteria defined in agreement with Bear Creek Mines. The Storage should be stacked every 10 m in height and intermediate stools with setbacks of 7.5 m between layers. Maximum height of the storage pile is 24 m. The base of the deposit will be composed of a layer of non-acid generating waste rock deposited during stripping operations until April 2021 and from May 2021 process tailings together with the mine waste rock will be deposited. The stacking ramps were designed with a total width of approximately 29 m and a maximum slope of 10%, based on the mine trucks operating parameters.

The filtered tailings and waste rock deposit is a large structure that will store mine waste and tailings, reaching a maximum height of 320 m on a complex topography. The installation of geotechnical instrumentation in certain areas, in order to monitor the displacements, and groundwater level in the structure, and also accelerographic stations to understand the seismic response in certain areas before a seismic event will be necessary. With the proper instrumentation, the displacements and variations in the conditions must be evaluated during the operation stage

Nineteen disposal stages, including the starting platform, beginning in year 2021 and through to 2039, in accordance with the mine plan for life of the project provided by BCM in June 2017 are foreseen. The arrangement during the stacking of the deposit considers the formation of the starting platform (with NAG mine stripping material) corresponding to the production of part of 2021, a stripping layer NAG with a minimum thickness of 10 m on the base surface, a thickness of at least 20 m of NAG waste on the exposed side of the deposit and a NAG stripping layer with a minimum thickness of 1 m at the top of the final configuration of the deposit to provide an encapsulation of the PAG waste rock. The interposed disposal of stripped and filtered waste will allow drainage of excess water from the tailings through the waste rock channels. To provide continuity to maximize the drainage during the disposal of the waste rock, windows must be left in a random manner, leaving isolated areas where no waste will be placed, thus facilitating the drainage of excess water from tailings to the base of the deposit.

Figure 18-9: Contact Water Pond and Freshwater Pond



19 Market Studies and Contracts

19.1 Concentrate Marketing

19.1.1 Markets

The project will produce a lead concentrate, containing the majority of the recovered silver as well as a separate zinc concentrate. The high silver grade of the lead concentrate will make it a desirable concentrate for smelters. The concentrates will be sold and shipped to Asian smelters.

19.1.2 Contracts

There are no established contracts for the sale of concentrate currently in place for this project.

19.1.3 Concentrate Transport Logistics

Concentrate will be shipped in bulk from the mine site to the port of Matarani at an estimated charge of US$72.36/wmt. This estimated cost was supplied by Penfold Limited within the “Corani Sales and Marketing Study,” June 2017.

19.1.3.1 Concentrate Transport Insurance

Insurance will be applied to the provisional invoice value of the concentrate to cover transport from the mine site to the smelter.

19.1.3.2 Owner’s Representation

An Owner’s representation will be applied to the provisional invoice value of the concentrate to cover attendance during unloading at the smelter, supervising the taking of samples for assaying, and determining moisture content.

19.1.3.3 Transportation Options

The decision was made to transport concentrate by container trucks to the Peruvian Port of Matarini. The container trucks will meet all required environmental regulations and are fully enclosed. The first five years of lead and zinc concentrate production will be approximately 110,000 mtpy and 80,000 mtpy, respectively, dropping down to an average of 70,000 mtpy for lead concentrate and 45,000 mtpy for zinc concentrate for the remaining 13 years of the mine life.

For this study, it is assumed that the concentrate will be sold to the Asian market. The transportation cost is estimated at US$32.16 per ton for shipping to the smelter by ocean freight, plus an additional port handling charge of $15.10. The Matarini port has authorization for handling zinc concentrates in bulk; however, authorization does not exist for lead concentrates. Therefore, the lead concentrate will need to be shipped in super sacks and containers. The combined shipping and handling price for container and bulk concentrates is currently very similar, within less than $1.00/wmt.

The total freight cost including overland freight plus ocean freight and handling fees for concentrates shipped from the Corani plant to the smelter is US$119.62/ wet metric ton.

19.1.4 Smelter Terms

Smelter terms and penalties were supported by an independent market analysis (Penfold, 2017). The data from this study were used to evaluate the revenue, charges, premiums, and penalties that are presented in detail below.

19.1.4.1 *Sale of Lead and Zinc Concentrates*

Every smelter has different rates for impurities depending on the normal feed. Higher levels of impurities will decrease the value of concentrates delivered to the smelter.

19.1.4.2 *Zinc Treatment Cost and Premiums*

Spot zinc treatment charges are half annual terms. Zinc metal concentrates are in demand due to shortages, and, as a result, the spot treatment charge has been reduced inversely to the Zinc LME price.

19.1.4.3 *Lead Treatment Cost and Premiums*

The price of lead has been fairly constant over the last several years and appears to be stable for the next few years. Table 19-1 shows the assays for materials found in each type of concentrate for this project.

Table 19-1: Concentrate Assays

Material	Units	Amount in Zinc Concentrate	Amount in Lead Concentrate
Ag	g/t	331.0	1815
Pb	%	4.22	58.9
Zn	%	53.8	5.0
Cu	%	0.41	1.60
Au	g/t	0.25	0.33
S	%	30.1	20.8
C(t)	%	0.18	1.02
Cl	g/t	34	<10
Hg	g/t	55.7	16.6
As	g/t	520	1215
Ba	g/t	338	981
Ca	g/t	1321	460
Cd	g/t	3499	1271
Fe	%	4.5	7.5
Sb	%	0.22	0.82
SiO_2	%	5.2	3.6

Additional test work will be performed to investigate lowering the quantities of some penalty elements; for example, reducing the amount of SiO2 in the Zinc concentrate as well as As and Sb in both concentrates. Lowering these impurities in the final concentrate will increase the value of the concentrate to potential smelters.

19.1.5 Sale of Concentrates

19.1.5.1 *Zinc Concentrates*

Typical Terms for the sale of zinc concentrates in the Far East as reported by Penfold Limited are as follows:

- Payable Metals
  - Zinc: 85 percent (minimum deduction of 8.0 units)
  - Silver: Deduct 3.0 ounces per Dry Metric Tonne (DMT) and pay for 74% of the balance
- Treatment Charge

- US$145/DMT Cost Insurance & Freight Free Out (CIF FO) Main Asian Ports basis a Zinc price of US$2,500 per MT and shall be increased / decreased for each US$1.00/MT off variance above or below US$2,500 per MT as follows:
  - Base T/C-$145.00 @$2,500 +6.0/-2.0 cents per dollar
  - Scale US$/MT

- Penalties
  - Fe: 8.0% free, $2 per 1% thereafter
  - As: 0.2% free, $2 per 0.1% thereafter
  - SiO_2: 3.0% free, $2 per 1% thereafter
  - Hg: 50 ppm free, $2 per 10ppm thereafter
  - MgO: 0.35% free, $1 per 0.1% thereafter
  - Bi: 0.1% free, $0.5 per 0.01% thereafter
  - Sb: 0.1% free, $1 per 0.1% thereafter
  - Co: 0.1% free, $1.75 per 0.01% thereafter
  - Mn: 0.4% free, $1 per 0.1% thereafter
  - F: 300 ppm free, $2 per 100 ppm thereafter
  - Cu+Pb – 4% combined free, $2 per 1% thereafter

19.1.5.2 Lead Concentrates

Typical Terms for the sale of Lead-Silver concentrates in the Far East as reported by Penfold Limited are as follows:

- Payable Metals
  - Lead: 95 percent (minimum deduction of 3 units)
  - Silver: 95 percent (minimum deduction 50 grams per DMT)
- Treatment Charge
  - US$150 per DMT
  - Refining Charge – Silver, $1.00 per payable oz.
- Penalties
  - As – 0.2% free, $2 per each 0.1% thereafter
  - Sb – 0.2% free, $1.5 per 0.1% thereafter
  - Bi – 0.1% free, $2 per 0.1% thereafter
  - Hg – 50 ppm free, $2 per 10 ppm thereafter
- Penalty element assay values were based on chemical analyzes of concentrates produced during the composite locked-cycle test performed by SGS.

19.1.5.3 Metal Prices for Study

The metals prices used for this study were as listed in Table 19-2.

Table 19-2: Metals Prices Used for Study

Material	
Zinc	US$1.10/lb
Lead	US$0.95/lb
Silver	US$18/troy oz

Due to size and remote location of this deposit it was felt a longer period of metal pricing should be considered, beyond the 3-Year trailing average. Base metal deposits by their nature require long lead times, and the current price may have limited relevance to the price when the project is constructed. For this reason, it was determined that combining a 3-Year trailing price average with a 3-Year forward price would have the most relevance.

BMO Capital Markets supplied a “Street Consensus Commodity Estimates” for BCMC as of July 2017. BMO utilized 35 sources, 32 and 29 for silver, zinc and lead estimated future pricing, respectively.

Penfold Limited completed the “Corani Sales and Marketing Study” as of June 2017 for BCM. The study was a comprehensive review and synopsis of the lead-silver concentrates market, zinc concentrates market, and logistics costs. In addition, pro forma concentrate sales invoices were provided as well as metal price forecasts. Price forecasts for the three commodities were from recognized banking institutions, metal brokers, and market analysts. Penfold credited 35 sources for silver future pricing, 28 for zinc, and 24 for lead.

The results from the BMO Capital information and the Penfold Study were averaged to generate a 3-Year Forward Average Commodity price. The 3-Year Trailing Average Commodity price was based on London Metal Exchange closing prices. The 6-Year Average within Table 19-3 is the average of the 3-Year Trailing and 3-Year Forward pricing. The Base Case Pricing represents the prices used for the base case economic evaluation.

Table 19-3: Commodity Pricing

Commodity	3-Year Trailing Average	3-Year Forward Average	6-Year Average	Base Case Pricing
Silver	\$16.63	\$19.68	\$18.16	\$18.00
Lead	\$0.88	\$1.00	\$0.94	\$0.95
Zinc	\$1.00	\$1.18	\$1.09	\$1.10

Base case pricing compares favourably with current spot pricing, (August month-end 2017); lead and zinc pricing are both well below current spot prices. Silver is slightly above the August month-end spot price by approximately 3%. The pricing used in this study is felt to be reasonable and rational by GRE.

20 Environmental Studies, Permitting and Social or Community Impact

Principal environmental risks associated with this type of project fall into these categories:

- Potential risks to air quality from dust;
- Potential degradation of surface and groundwater quality;
- Potential changes to the volume of surface and groundwater;
- Visual impacts due to the creation of pits, mine waste disposal facilities, roads, and other mine workings; and
- Permanent changes to land use resulting from mining activities.

The existing Environmental and Social Impact Assessment (ESIA) quantifies the magnitude, extent, and mitigation of risks and potential impacts related to the old project configuration which included a conventional wet tailings facility and related water supply infrastructure. The new project configuration developed for the current study will be incorporated into a modification of the ESIA to be completed subsequent to the publication of this report. Project optimization performed for the current study resulted in a reduced project footprint, a reduction in water consumption, and other changes which are anticipated to reduce environmental impacts associated with project development relative to the previous study. In a number of cases, the development of the Project is anticipated to improve existing environmental conditions.

Peruvian Law 28090 regulates the obligations and procedures mine owners must follow in relation to mine closure, and requires that a mine closure plan be approved and financial guarantee for the cost of implementation be established. The plan must describe the rehabilitation methods and their costs for the operation, closure, and post-closure phases. The plan must allow for progressive closure, with mine owners reporting semiannually to the Ministry of Energy and Mines on progress with the implementation of the approved plan.

As required under Peruvian regulations, a preliminary closure plan was submitted and later approved in September 2014 (Walsh, 2014). A revised closure plan based on the current project configuration will be submitted to Peruvian regulators following modification of the ESIA .

The development of closure concepts for this study has considered International Finance (IFC) guidelines and industry standards in addition to Peruvian regulatory requirements. The general approach to mine reclamation and closure developed at this time is described below. The estimated cost (based on the Preliminary Closure Plan) has been incorporated into the overall cost estimate for the Project.

The sections below present the following:

- A summary of existing studies;
- Known permitting considerations;
- A brief description of the strategies for environmental management during operations, reclamation and closure;
- Socioeconomic and community considerations.

20.1 Environmental Baseline Studies

Environmental sampling has been ongoing since July 2009. Extensive additional site characterization was conducted in 2011 and the first half of 2012; the results of these studies were presented in the ESIA (AMEC, 2012). The following sections summarize the existing environmental data.

20.1.1 Summary of Air, Noise, Groundwater and Surface Water Studies

Below is a summary of the results from the currently-published baseline studies for air, noise, and water:

- The air quality results were below the maximum permissible limits set by the national environmental standard for air, reflecting the absence of significant airborne pollutant-generating activities in the zone.
- Noise measurements were taken near populated areas. The results were below the maximum thresholds specified in the national environmental standard for residential zones.
- Groundwater appears to be located in shallow aquifers comprised of alluvial, glacial, and aeolian deposits that lie over low-permeability basement rocks. Results from the site investigation indicated that the shallow aquifers have high storage but moderate to low hydraulic conductivity. Little evidence exists for a conductive and extensive hard-rock aquifer. In general, it appears that groundwater resources in the Project area are not sufficiently large to be useful for agricultural, domestic, or industrial use. However, the protection of the groundwater resources from impact is part of the focus of the ESIA and future environmental planning.
- Surface water samples exhibited highly to slightly acidic characteristics during sampling events conducted throughout the year. These acidic conditions are related to naturally-occurring oxidation of mineralized rocks exposed at the site and from areas previously disturbed by historic mining activities. Several metal concentrations exceeded the national environmental water standards. Similarly, some metal concentrations measured in sediments exceeded Canadian Environmental Quality Guidelines. As a result, the major drainages leaving the Project area do not currently meet national environmental water standards.

20.1.2 Summary of Biological Studies

The biological baseline study describes the ecosystem of the site and the species abundance, richness, biodiversity, and endemism. The species present on the site have been cross-referenced with threatened species lists (international and national). Of particular note is that:

- A number of flora are included on the national threatened species list but not on international conservation lists, including umbellifer, certain daisy species, and Valerina nivalis.
- Three vegetation species found are considered endemic to Peru, of which Nototriche pelicea is considered endemic to the Department of Puno.
- A number of mammal species identified in the biological baseline study are included in both national and/or international threatened species lists, including Hippocamelus antisensis (north Andean deer), Leopardus colocolo (pampus cat), Puma concolor (puma) and Vicugna vicugna (vicuña). However only Vicugna was observed in the field. The remaining species were recorded based on sightings from members of the local community; therefore, their presence on the site is not confirmed.

20.1.3 Summary of Geochemical Studies

A geochemical waste characterization program was developed to assess the acid rock drainage (ARD) behavior of, and potential leaching of contained metals from, all mine wastes associated with the Project. This program included static tests, LECO Furnace total sulfur and total carbon assays, and onsite and laboratory kinetic cell tests, as presented in the ESIA (AMEC, 2012).

These were combined with geologic and metallurgical characterization of lithologies and material types. The conclusions from this work included the following:

- The geochemistry of the waste rock will be dominated by certain mineralization types; in particular, mineralized lithic tuff with fine black sulphides (FBS) and mineralized tuff with pyrite and marcasite (PM).
- Whole rock analysis of waste rock samples indicated that several metals of environmental and processing concern exist at high levels; synthetic precipitation leaching procedure testing suggested that many of these metals are readily leachable.
- The kinetic tests showed that many waste types were acid generating, though the behavior among certain mineralization types was mixed.
- The ABA indicator testing of the tailings suggested that the tailings would be potentially acid generating (PAG) material due to the presence of residual sulfides.

Additional geochemical humidity cell kinetic tests were initiated in 2014 by Amphos 21 (Amphos 21 Consulting Peru, SAC, 2014). Initial results from these tests confirmed previous kinetic cell testing by demonstrating that the PMT is non-acid generating, and that sulfidized waste often produces acidic leachate.

20.2 Permitting Considerations

Refer to Section 4.4 for an explanation of the permits required to execute the Project.

20.3 Environmental Management During Operations, Reclamation, and Closure

The project design approach considers implementation of best management practices and a sustainable approach to project development. The project development approach has incorporated the ability to close or reclaim project facilities progressively throughout the project life. A more detailed description of the potential impacts associated with the project development can be found in the ESIA and Closure Plan studies.

20.3.1 Environmental Management Objectives

The operational environmental management objectives are to identify potential and viable measures to mitigate environmental impacts that can be implemented during the operation, reclamation, closure, and post-closure periods. These measures are intended to alleviate potential long-term impacts from mining operations and to minimize long-term liability. Current conditions at the Project site show a degree of degradation of water quality in the general vicinity of the mineralized zones. This is due to naturally-occurring oxidation of sulfide-bearing materials, which has resulted in a depression of pH and an increase in dissolved metals and salts in surface waters leaving the Project area. In addition, historic mining activity within the main Corani basin has resulted in the presence of underground mine workings, waste rock, and historic mine tailings which are a significant source of ARD.

The measures to be applied for reclamation and closure of the Project are intended to return the receiving environment to a condition that is, at worst, equal to the measured baseline conditions, and to the extent practical, better than these conditions. The closure measures are not intended to change the naturally occurring conditions at the site, nor are they intended to mitigate the effects of historic mining within the project area, apart from where future mining activity can be used to implement mitigation measures in the course of the proposed mining, processing, and mine waste management defined for the Project. Historic mine areas will be closed/mined out in the pit areas and areas directly in the footprint of the Main Dump. Historic mine workings that are not directly affected by mining activity, but are still within the catchment area of the mine’s water management facilities and are contributing ARD to the facilities will also be closed. Moreover, it should be noted that although a large number of the historic mining facilities contributing to ARD will be removed during Corani mine operations and closure, a significant portion of the undisturbed bedrock at the site is naturally acid-generating, and will continue to produce ARD during operations and closure.

The following activities performed during operations, reclamation and closure are intended to mitigate and minimize generation of ARD:

- Encapsulation of PAG tailings and waste rock within the Main Dump;
- Extraction of legacy mine waste and disposal within PAG encapsulation cells inside the Main Dump;
- Saturation and encapsulation of PAG tailings and waste rock within the pit backfill areas;
- Concurrent placement of reclamation covers over the Main Dump and the Este backfill during operations;
- Segregation of contact waters from non-contact waters and consumption of contact water, and a portion of naturally acidified runoff from the watershed, in the flotation plant during operations;
- Consumption of all contact water by the plant.
- Backfilling of the Minas and Main pits to prevent formation of pit lakes and placement of an ET cover.

In the following sections, the overall site conditions that affect the selection and implementation of the reclamation and closure measures are identified. The reclamation and closure considerations for individual project components and facilities are then presented. The general items considered in estimation of costs for the reclamation, closure, and post-closure periods are also identified. A large proportion of the reclamation and closure measures will be completed as part of operation of the mine. Cost estimation for reclamation and closure includes only those activities required following the end of mining and mineral processing. For example, the cost for pit backfilling does not include the cost for backfilling that occurs during the course of normal operations which is covered under mining costs. Additional information regarding the cost estimation for specific items is provided in the relevant sections of this document.

20.3.2 General Site Conditions Relevant to Environmental Management

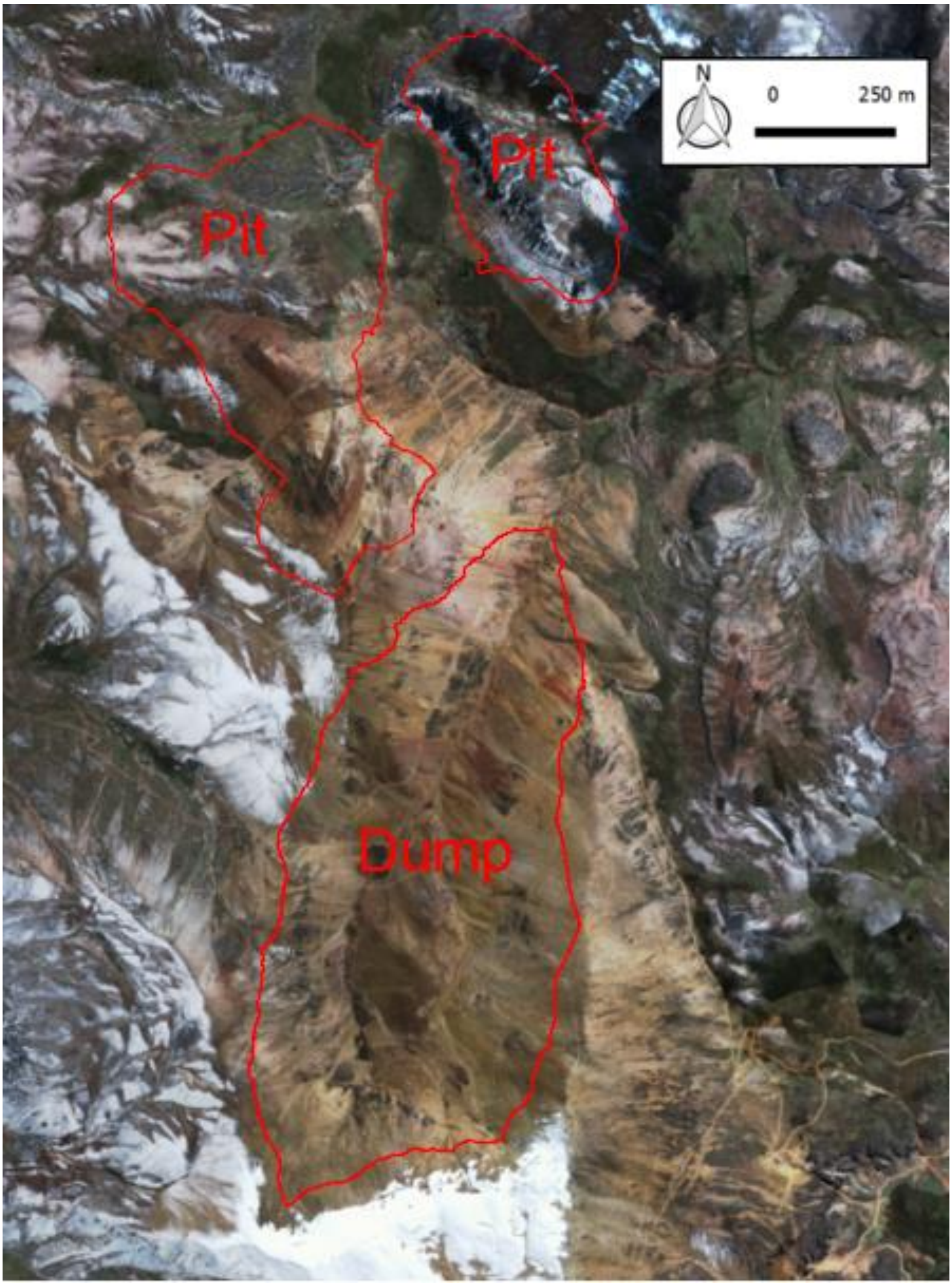
As described in Section 7 of this document, some areas of the project experience naturally-occurring ARD. Consequently, in these areas, naturally occurring growth media are limited or absent. Key areas without soils capable of sustaining vegetation include:

- The valley that will contain the Main Dump and portions of the plant complex;
- Large portions of the mountain slopes in and around orebody; and
- Higher elevations of the project site and areas with exposed bedrock.

The areas not capable of sustaining vegetation are readily apparent from the color aerial photo shown in Figure 20-1. It is important to note that many of these areas will not sustain vegetation post-closure.

The reclamation and closure measures identified in the following subsections have been developed to minimize post-closure management requirements for the areas of disturbance related to mining activities. However, as described further below, it is anticipated that some long-term post-closure management and monitoring will be required for an extended period following mine reclamation and closure.

Figure 20-1: Aerial Photograph of the Corani Site, Current Conditions



20.3.3 Project Components and Facilities

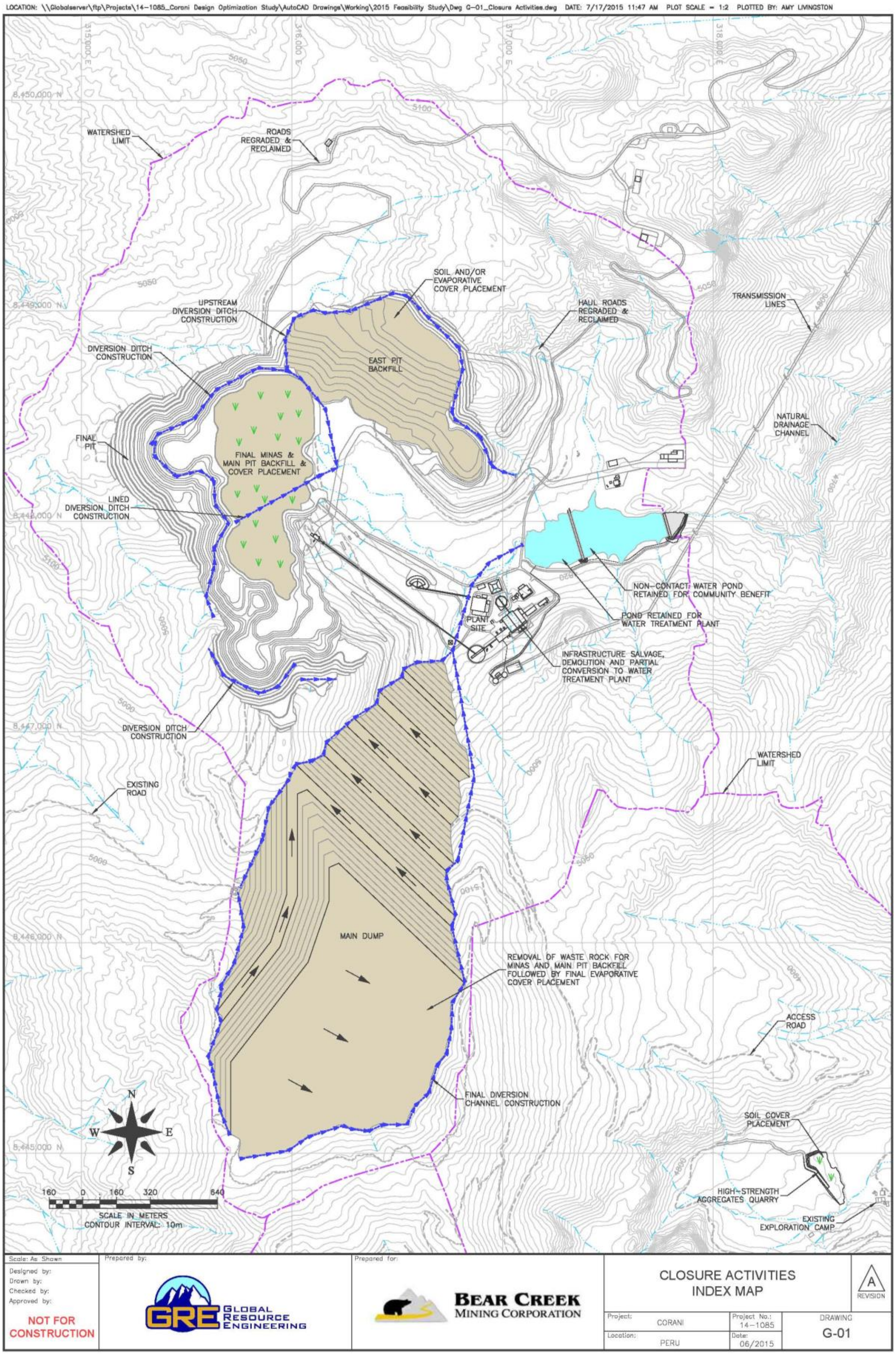
The following sections describe the environmental management plans for specific project components and facilities. Additional design details can be found in Section 18. The main project components and facilities with environmental management and/or reclamation considerations include the following:

- The Este, Minas and Main Corani open pit areas;
- Waste rock and tailings co-disposal areas included in the Main Dump and in-pit backfills;
- The plant facilities, crushing facilities, tailings dewatering plant, and related infrastructure;
- The surface water management infrastructure and Plant Contact Water Pond, and the Plant Non-Contact water pond;
- Soil stockpiles areas;

It is not the intent to identify all aspects of environmental management, reclamation and closure measures at this time. Rather, only the major aspects with significant cost implications are considered.

The project configuration at the end of mining, just prior to closure, is shown in Figure 20-2 shows the preliminary closure plan concepts described in the following sections.

Figure 20-2: Preliminary Closure Plan Concepts



20.3.4 Open Pits

Mining of the open pit will occur over a period of approximately 18 years. Upon completion of mining, the Este, Minas, and Main pit areas will form a discontinuous horseshoe-shaped excavation around the east, north and west sides of the Corani valley.

20.3.4.1 Bodedal Soils

Since the 2015 FS, the pits have been redesigned to avoid disturbing bofedal soils. However, localized disturbance may occur.

Localized stockpiles of bofedal material will be established for use in reclamation and closure activities. Segregation of the materials will be performed to separate organic material from the rest of the excavated soils and unconsolidated materials. Collection and storage of these materials will be maximized to the extent practical.

20.3.4.2 Pit Water Management

Several natural drainages will be tributary to the fully developed pit area, and these drainages will continue to contribute surface runoff toward the pit. During operations, this water will be diverted around the pit or routed into the channel that is retained between the Este and Minas pits. This water will be classified as non-contact and can be discharged to the environment, if desired. However, runoff generated within the footprint of the pits and groundwater inflow to the pit will be collected in sumps and pumped to contact water pond for use as make-up water.

The goal of the surface water management system at closure will be to reduce the volume of water requiring treatment by separating water that has been in contact with potentially acid generating material from water that has been in contact with only non-acid generating material. Water that has been in contact with potentially acid generating material (runoff from exposed PAG pit slopes and seepage emerging from the pit backfill) will be piped to a water treatment plant located at the former plant site. This water will be treated as necessary before being released downstream. It is anticipated that a High-Density Sludge (HDS) water treatment and neutralization system will be employed.

Water from the areas up-gradient of the pit will be diverted to drop structures that will convey the water over the exposed pit wall, and then routed over the pit backfill areas in lined channels or in the channel between Este and Minas pits, before release to natural drainages downstream. Diversions will be built to minimize runoff from non-acid generating portions of the pit wall. It is anticipated that this water will not require treatment and therefore will be routed around acid-generating portions of the pit wall and released downstream.

20.3.4.3 Backfill

The Corani pits will be backfilled for the purpose of storing waste rock and tailings, and also to prevent formation of pit lakes which could contain acidic water with elevated dissolved metal and salt constituents. Concurrent reclamation and closure of the mine pit will be performed as feasible during mine operations and completed at closure.

Backfilling of the Este pit will take place during mine operations using tailings and waste rock generated by ongoing mining of the Minas and Main pits during that time. Tailings and waste rock will be placed in designated backfill zones. After backfilling has been completed, an engineered soil cover will be installed over the backfilled area. The cover will consist of a layer of NAG material overlain by topsoil or bofedal soils and will limit the amount of precipitation water that can enter the backfill thereafter.

Mining in both Minas and Main pits prevents backfilling of this pit area until the end of the mine life. Backfilling of the Minas/Main pit area will occur over the course of several months directly following completion of mining. The area will be backfilled by importing waste rock and tailings temporarily placed in the Main Dump. Backfill will be placed to a level high enough to ensure that the final backfilled surface remains above the water table after hydrologic stabilization occurs during the post-closure period.

After this material has been placed, available water will be diverted to artificially recharge the backfill, saturating the material as quickly as possible, and limiting the time during which the backfill and the pit walls are exposed to atmospheric oxygen. Artificial recharge of the Este backfill is not required due to the low hydraulic conductivity of the tailings backfill.

The final Minas/Main backfill surface will be graded to resemble the topography of the adjoining lower bofedal area, and to merge into that topography. Stockpiled topsoil will be used to cover the backfill, with the intention of promoting gradual natural establishment of a vegetated environment that is an extension of the lower bofedal. A preliminary model of the cover was developed using a numerical model to evaluate the performance of various engineered cover configurations. Based upon the model results, direct infiltration through the cover is expected to be reduced to a minimal level following cover installation.

A hydrogeological model was developed to simulate the groundwater flow through the backfill after flooding of the Minas/Main backfill and installation of the surface water management and cover systems. The model results indicate a relatively small seepage rate emerging from the backfill. Seepage emerging from the backfill will be collected and routed to the treatment plant as necessary.

20.3.5 Main Dump

The Main Dump is described in Section 18; this section covers the reclamation of the main dump.

During operations, dump slopes will be progressively reclaimed with the placement of previously stockpiled natural soils from the base of the dump. Runoff from these areas will be routed around the active dump areas and released downstream to natural drainages. Following the placement of the reclamation cover, runoff from the reclaimed surface will be directed to toe ditches and routed beyond the facility. At closure, any seepage from beneath the toe of the dump will be collected and treated as necessary prior to release.

The current condition of the ground in Quebrada Muerta beneath the Main Dump is un-vegetated and sterile. The goal of the closure plan is to establish conditions similar to those that existed prior to mining activities, and therefore, does not include the establishment of vegetation on the Main Dump. The closure plan includes erosion control methods in place of vegetation.

20.3.6 Plant Facilities & Related Infrastructure

During operations, runoff generated in the area of the plant footprint is considered contact water and will be diverted to a sump and pumped to the contact water pond for use as makeup water. As part of reclamation and closure activities, a portion of the mineral process plant will be converted to a water treatment plant. Water from the surface water management system will be routed to the plant, and treated as necessary prior to release to the receiving environment with a high-density sludge (HDS) treatment system.

At the end of the operating life of the plant, with the exception of those components to be utilized in the post-closure water treatment system, the plant will be decommissioned and demolished. Where viable, equipment will be salvaged, sold, or transported offsite as scrap material. A small amount of infrastructure will be left in place for post-closure use. This would include a water treatment and equipment storage area, the office building, and equipment storage required for the maintenance and monitoring of post-closure site conditions. All other structures will be demolished, and the demolition debris buried in an on-site landfill. This would include the majority of the plant buildings, truck shop, explosives magazine, crushing system, warehouses, reagent storage areas, etc. The main power line utilized during operations is retained for the HDS treatment plant. All major pipelines will be either salvaged or, if buried, left in place.

Concrete foundations for the plant will be broken up and buried in place in most areas. The degree to which they are broken up will be determined by the ability to create a natural-appearing post-mining topography with a natural drainage pattern. Leaving foundations in place may be considered where this can be accomplished without breakage. A suitable soil cover will be placed over these areas to facilitate the function and appearance of natural ground.

Once the off-site man camp is no longer required, the camp will be converted to an alternative beneficial use by a future custodian and/or local community representative. The main access road and haul roads left at the end of mining operations that will be required to provide access to the site, and access to locations necessary for maintenance and monitoring of the project areas, will be reduced in size to that of similar local roads in the project region. A portion of the internal roads will be removed by ripping, and reclaimed to conditions similar to the surrounding area; the remainder of the roads will be converted to community use.

20.3.7 Site Wide Water Balance

A project wide water balance was developed for the operational mine life as part of project design. The water balance was created to satisfy the following requirements;

- To prove that sufficient water exists to run the plant, even during extreme dry conditions;
- To prove that contact water can be consume by the operation, thus alleviating the need for water treatment and discharge;
- To ensure that sufficient non-contact water exists to maintain environmental baseflow conditions in the Chacaconiza drainage basin downstream from the Project.

The water balance considers the inflows and outflows to each Project area over the operational mine life. These flows are varied over the life of the project on a monthly basis to determine balance of contact and non-contact water. This variation also occurs as facilities are operated over time. For example, as the pit increases in size, the amount of contact water increases. In contrast, following the ramp-up period, the annual process consumptive use of water remains steady over the life of the mine.

In accordance with industry best-practice, a probabilistic water balance has been created. The model determines the probable ranges of conditions that may exist during operations. The model automatically varies a number of multiple climate input parameters, (such as precipitation, snowfall and evaporation) based on a statistical analysis of baseline historic data. The resultant possible combinations form the basis of a stochastic, probabilistic model built in the GoldSim modeling platform. GoldSim creates a stochastic dataset of probabilistic model iteration, and from these iterations, one is able to select a range of design criteria. For this study, the 95% wet and the 5% dry conditions are used as the primary criteria for water management infrastructure.

A water balance is a dynamic tool for project planning. Section 26 outlines additional analysis necessary prior to project construction.

The water balance model is based on climate parameters modeled by Amphos 21. Precipitation inflows are based on a one thousand stochastic iterations of a synthetic precipitation series that include both the 100-year wet storm and the 100-year dry storm. Modeling the project under 1,000 different climate combinations simulates the natural variability that would be expected over the project life.

The database used for this analysis comes from a site weather station that has been recording hourly data since December 2008, and flume stream gauging stations that have been recording hourly data at the site since 2012. Site-specific runoff coefficients were calculated using the combined data record available from 2012 to 2017. Incorporation of the site-specific data measurements offers a significant improvement over previous studies, which relied on literature based assumptions for runoff coefficients. Groundwater inflows to the mine pits were estimated based on a transient MODFLOW groundwater model reflecting the mine plan and pit backfilling schedule.

During Project development and operations, a surface water management system will be developed (see Section 18) to route runoff from areas of disturbance, groundwater collected in the pit, and seepage from the Main Dump to the contact water pond and then to the process plant for use as makeup water. The capacity of the collection pond is approximately 830,000 m^3, and will consist of two separate cells: a non-contact water cell (in the downstream portion, with a volume of 572,000 m^3) and a contact water cell (in the upstream portion, with a volume of 257,000 m^3). Both contact water and non-contact water stored in the ponds will be used to supply the plant during the dry season, as required.

Environmental baseflow in the Chacaconiza drainage will be maintained during Project development subjected to the natural availability of flow in the catchment area. Concepts related to the surface water management plan are presented in more detail in Section 18.

The total process water demand is discussed in Section 18. The process plant makeup water demand is satisfied from water available in the watershed, with the following water supply priority order: contact water (runoff and baseflow from disturbed areas and pit inflow), non-contact water (runoff and baseflow from undisturbed areas in the catchment), water from the upstream cell of the Plant Water Pond (contact water and stored non-contact water), and finally, water from the downstream cell of the Plant Water Pond (stored non-contact water). Once the plant demand is met, the excess non-contact water from the remaining area(s) is either stored in the Plant Water Pond or released downstream. Under the 95% wet condition (see above), all contact water is consumed by the plant or temporarily stored in the upstream cell of the Plant Water Pond and is not discharged to the environment. Conversely, the combined pond contains sufficient water to supply the plant under the 5% dry condition (see above).

Figure 20-3 displays the contact water pond storage volume under all modeled probabilities and Figure 20-4 displays the non-contact water pond storage volume under all modeled probabilities. The black line in the chart displays average storage conditions and the 5th and 95th percentile results -- the extreme dry and the extreme wet condition -- , respectively.

Figure 20-3 Contact Water Pond Storage

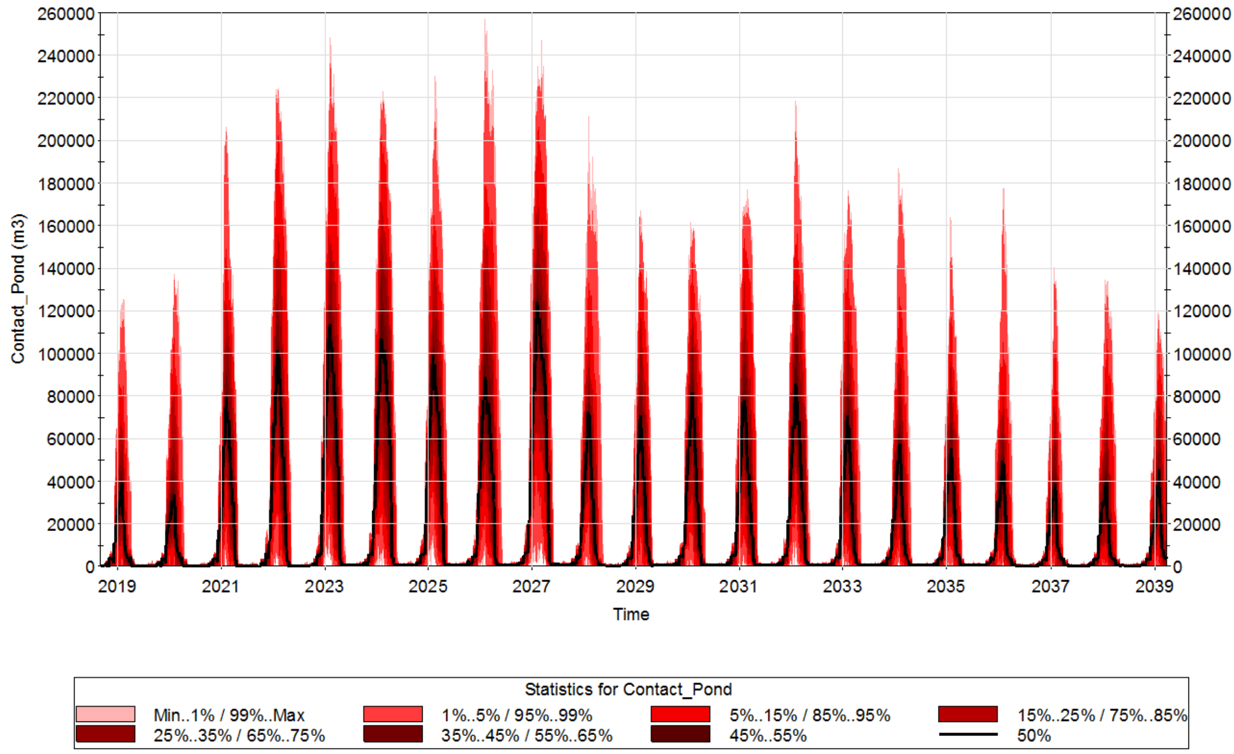
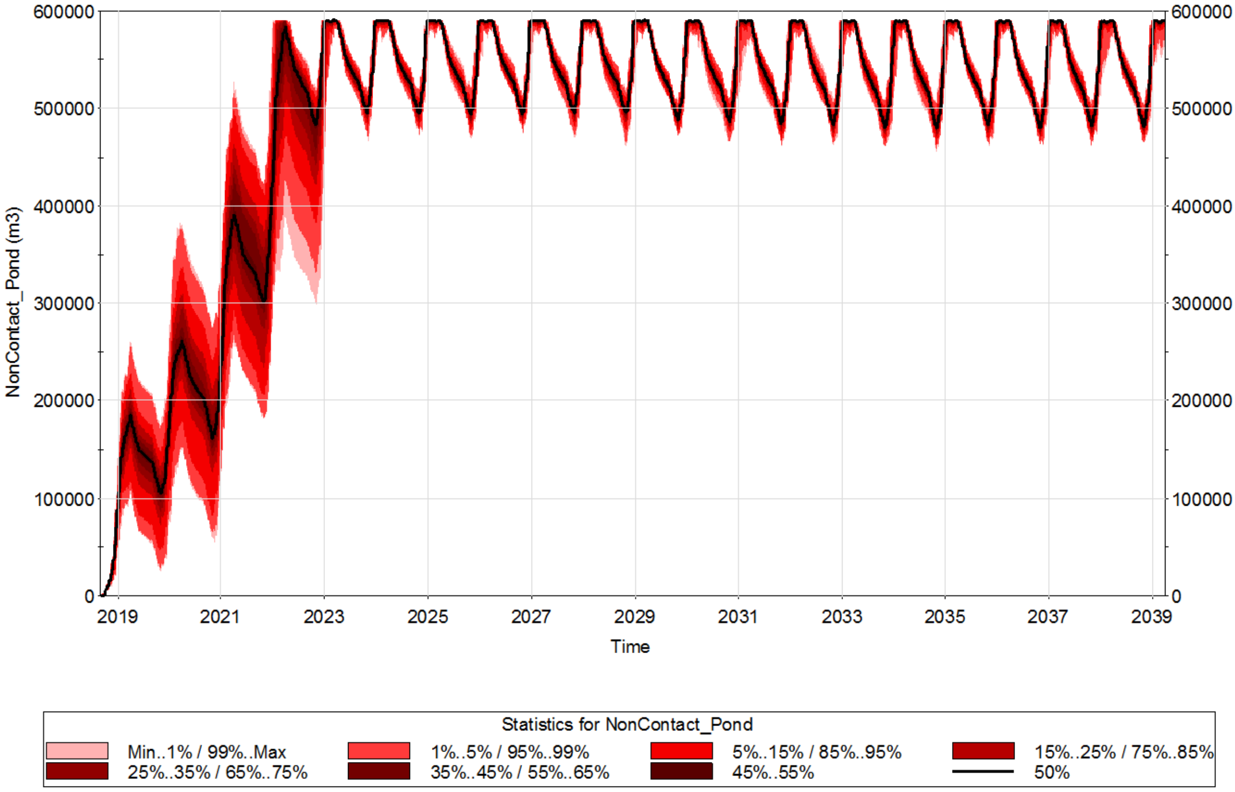


Figure 20-4 Non-Contact Water Pond Storage



The water available in the watershed greatly exceeds what is required for operations on an annual basis. This ensures that the non-contact water pond is never dry, and that environmental baseflow can be maintained, even under extreme conditions. Conversely, the project has sufficient consumptive requirements and storage capacity to consume all contact water.

20.3.8 Closure Phase Water Management

The contact water cell of the plant water pond will be used during closure as a flow equalization pond for the water treatment plant. Use of the pond will be used in the first year of closure to store ARD-impacted water while the process plant is converted into an ARD treatment plant.

At closure, the non-contact pond could constitute a potential resource for the downstream communities. Transfer of ownership and operation of the pond to a group of the local communities or government organizations would be performed following the implementation of reclamation and closure measures.

Upon closure, permanent water conveyance structures will be retrofitted as required for minimum maintenance. This may include the installation of new riprap channels and passive sediment control structures.

20.3.9 Regrading Plan

Elements of the generalized regrading and re-vegetation plan include:

- All visibly contaminated soils will be excavated and treated in an on-site soil land farm;
- All compacted soils will be tilled with a disc to relieve soil compaction and improve drainage;
- Impacted areas will receive a soil cover, where cover existed prior to development;
- Impacted areas will be revegetated with native species, where capable of supporting vegetation; and

- Erosion-control BMPs will be applied to minimize erosion until revegetation is established.

If required, impacted soil will be disposed of in a hazardous waste facility, but experience with other mining operations suggests that the potential quantity of soil that cannot be land-farmed will be small.

20.3.10 Monitoring and Maintenance

Monitoring and maintenance during operations will ensure that the control measures used to prevent environmental impacts are effective. This will include climate monitoring, water quality sampling, groundwater and surface water monitoring, air monitoring, and geotechnical monitoring of the Main Dump, pits, and pit backfill. The monitoring program for operations is described in more detail in the ESIA.

Monitoring and maintenance of the mine reclamation and closure activities will begin while these activities are underway, and will continue in the post-closure period. Monitoring and maintenance will be done primarily by a team of onsite personnel. Equipment will be made available for the monitoring of reclaimed areas of the project and for the performance of regular maintenance. The monitoring will include water quality sampling and analysis, in addition to regular inspection of the Main Dump, process area, and the pit areas. The required duration of the monitoring and maintenance is undefined at this time, but a minimum period of 25 years has been considered in the cost estimate.

Many monitoring activities carried out during operations will be continued in closure, including:

- Automated weather station monitoring
- Manual water level measurement in monitoring wells
- Manual water quality sampling in monitoring wells
- Manual water quality sampling of the water treatment plant effluent
- Automated stream flow monitoring via permanent flumes
- Manual surface water quality sampling
- Automated geotechnical monitoring at the Main Dump and the Este Pit Backfill
- Site-wide inspections for revegetation success, erosion control, surface water management, and general maintenance.

Maintenance will also focus on the surface water channels around the pit, over the pit backfill, and the Main Dump. Sediment removal and repair of any damage to channel lining will be specific areas of focus for the maintenance program.

20.3.11 Closure Schedule

The schedule for performing reclamation and closure activities can be summarized as follows:

- Este pit backfill - Concurrent reclamation during operations.
- Minas/Main pit backfill - Completion of backfill reclamation three years after closure.
- Main Dump - Concurrent reclamation of completed surfaces as they become available, beginning in Year 2; completion of activities in tandem with the Minas/Main pit backfill.
- Surface water management system – As soon as possible after closure.
- Water treatment plant conversion - As soon as possible after closure

20.4 Socioeconomics and Community

The Project site is located within the jurisdiction of two rural communities (comunidades campesinas), Chacacuniza and Quelcaya. These communities, situated at over 4,000 masl, are small isolated rural settlements, with a population of around 120 individuals each. The majority of the population speaks Quechua as their first language.

The geographical isolation and lack of infrastructure in the zone result in reduced economic and social development opportunities, and high levels of poverty. However, both communities have access to water, electricity, and rudimentary wastewater services. The main housing construction material is adobe, with thatched roofs and earth floors. The most common illness is respiratory infections which children and the elderly are susceptible to during periods of snow and cold weather.

BCM has undertaken community relations activities as part of previous permitting activities related to exploration campaigns. Further activities were undertaken as part of the ESIA that was approved by the Peruvian Government in 2013. The ESIA process included a thorough description of the current social and economic status of the communities, and the analysis of possible and anticipated, positive and negative impacts on these communities. Following analysis of the impacts, methods were developed to avoid, remedy or mitigate the identified negative community impacts, including the development of social programs aimed at providing enhanced economic and social development opportunities. These are presented in the ESIA.

Further community engagement occurred concurrent with the impact assessment process. This included two initial workshops with each of the two communities (Chacacuniza and Quelcaya) affected by the project. The first workshops were undertaken before the ESIA studies began, and the ESIA process was explained to the community. Additional workshops were undertaken during the elaboration of the ESIA. Finally, a public meeting was held. At this meeting, the ESIA and Community Participation Plan were presented to the approving authorities. Since the mine plan described in this document results in a reduced Project footprint and a reduction in water consumption, it is anticipated that impacts to the community will be reduced relative to those described in the current ESIA. For this reason, the ESIA modification is expected to be considered a technical modification which does not require additional public hearings.

Additionally, Bear Creek completed a Life of Mine ("LOM") Investment Agreement in June 2013. This agreement was entered into with the District of Carabaya, five surrounding communities, and relevant, ancillary organizations specifying investment commitments over the 23 year project life, including the preproduction period. Under the agreement, annual payments are to be made into a trust designed to fund community projects totaling 4 million nuevos soles per year (approximately $1.6 million per year), beginning with the first installments payable in 2013. Payments will remain constant throughout the pre-development phase and during production. Cessation or interruptions of operations will cause a pro-rata decrease in the annual disbursements. As an integral part of the LOM agreement, a trust or foundation structure is established for approval of investments and disbursement of funds. Each of the five communities (Corani (Aconsaya), Chacacuniza, Quelcaya, Isivilla, and Aymana) has agreed to the formation of committees which will consider and approve investment projects for the benefit of the communities, such as schools, medical facilities, roads, or other infrastructure. The amounts of the total annual investment to be directed towards each community is agreed to and defined in the agreement. Bear Creek is an oversight member of the trust and will assist towards the success of the projects; however, the Company will have no voting powers. In this structure, Bear Creek's intent is to appoint independent members with community social responsibility experience and credibility in order to provide oversight of the foundation's functions in meeting its commitments to the communities and all of its members.

21 Capital and Operating Costs

Capital and operating costs for the Corani Project were based on the Phase 1 FEED mine plan and process plant design. The capital costs were based on estimates for the equipment, materials, labor, and services required to implement the design. Operating costs were based on estimates of labor, materials, power, supplies, fuel, and estimates from consultants and potential suppliers to operate the mine and plant as designed.

The capital and operating cost estimates have been prepared by Graña y Montero (GMI), and reviewed by GBM Minerals Engineering Consultants Limited (GBM). GBM's Managing Director Michael Short is acting as Chapter 0 Qualified Person.

21.1 Capital Cost Summary

The capital cost estimate (CAPEX) presented in this Report is for a silver-lead-zinc mine and concentrator capable of producing and processing an average of 22,500 tpd of ore (dry basis). The total life of mine capital investment for the Corani Project is estimated to be $585.559 million as shown in Table 21-1.

Table 21-1: Life of Mine Capital Cost Summary

Cost Type	Cost (USD Million)
Sunk Costs	4.279
Initial CAPEX	580.919
Sustaining CAPEX	0.361
TOTAL	**585.559**

The initial CAPEX includes the design, permitting, pre-stripping, construction, and commissioning of the mine, plant facilities, ancillary facilities, utilities and camp. The CAPEX also includes costs for engineering, construction management, and Owner's costs.

In Table 21-2 a summary of the total initial capital to be spent is presented by major area.

The primary assumptions used to develop the CAPEX are provided below:

- All cost estimates were developed and are reported in United States of America Dollars (USD or US Dollars).
- Qualified and experienced construction contractors will be available at the time of Project execution.
- Borrow sources are available within the Project boundary.
- There is no allowance for weather related delays in construction in the estimate.

An estimate of contingency has been made based on the accuracy and level of detail of the cost estimate. The purpose of the contingency provision is to make allowance for uncertain cost elements which are predicted to occur, but which are not included in the cost estimate. These cost elements include uncertainties concerning completeness and accuracy of material take-offs, accuracy of labor and material rates, accuracy of labor productivity expectations, and accuracy of equipment pricing.

Table 21-2: Capital Cost Summary

Cost Area	Cost (USD Million)
General	40.337
Mining	44.544
Infrastructure	56.813
Process Plant	245.059
Fresh Water/Process Water	41.549
Power Supply	2.934
Ancillary Buildings	22.977
Engineering	10.000
Project Management & Supervision	14.980
Commissioning, Start Up and Vendor Representatives	13.888
Owner Costs	32.259
Contingency & Escalation	55.579
TOTAL	**580.919**

21.1.1 Currency

The estimate is expressed in US Dollars, as at May 2017. A provision has been made to account for future escalation of approximately $14 million of Project costs. No additional funds have been allocated in the estimate for further escalation or to offset potential currency fluctuations.

21.1.2 Estimate Exclusions

Items not included in the capital estimate are as follows:

- Reclamation costs (included in the financial analysis).
- Escalation beyond that discussed in Section 21.1.1.
- Foreign currency exchange rate fluctuations.
- Interest and financing cost.
- General sales and withholding taxes (included in the financial analysis).
- Working capital is excluded from the estimate. There is an allowance for cash inventory in the financial analysis, and an allowance for spare parts and first fills has been included in the capital estimate start-up costs.

Risks due to political upheaval, government policy changes, labor disputes, permitting delays, weather delays or any other force majeure occurrences are also excluded.

Accuracy

The estimate has been developed to a level sufficient to assess/evaluate the Project concept, various development options and the overall Project viability. After inclusion of the recommended contingency, the capital cost estimate is considered to have a level of accuracy of ±15%.

21.2 General Capital Costs

General capital costs are shown in Table 21-3. These costs relate to the supply and installation of a number of workshops, the supply and installation of communications systems, including 3G cellular telephony, broadband internet, telephone lines and VHF radio, and undertaking site earthworks.

Table 21-3: General Capital Cost Estimate

Cost Type	Cost (USD Million)
Earthworks	27.374
Concrete	0.714
Structural	11.905
Instrumentation	0.344
TOTAL	**40.337**

21.3 Mine Capital Costs

Mining will be performed using contract mining. Initial mining capital costs are shown in Table 21-4 Mining capital costs are associated with establishing the haul roads, the ROM pad and capitalised operating costs associated with pre-stripping. The pre-production costs are sourced from a contract mining quote.

Table 21-4: Mining Capital Cost Estimate

Cost Area	Cost (USD Million)
Topography Equipment	0.042
Software	0.605
Dispatch System	0.361
Contract Preparation	0.250
Haul Roads	9.562
Phase 1 Construction Contractor Mining	10.621
Pre-Production – Mining	16.773
Pre-Production – Mining Contractor Fee	2.264
Indirect Costs	3.831
ROM Pad	0.235
TOTAL	**44.544**

21.4 Infrastructure Capital Costs

Infrastructure capital costs include the tailings deposit facility, internal roads, the main access road and improving existing access. These costs are shown in Table 21-5.

Table 21-5: Infrastructure Capital Cost Estimate

Infrastructure	Cost (USD Million)
Tailings Deposit Facility	8.015
Internal Roads	7.549
Improving Existing Access	1.811
Main Access Road	
Section 1: Huiquisa -Tantamaco - Isavilla (8.6 km)	20.961
Section 2: Isivilla Bypass (2.2 km)	1.327
Section 3: Isivilla - Machaycunca - Jarapampa (21.2 km)	7.629
Section 4: New Road (12.1 km)	9.521
TOTAL	**56.813**

21.5 Process Plant Capital Costs

The process plant capital costs are provided by process plant area in Table 21-6 and by discipline in Table 21-7.

Table 21-6: Process Plant Capital Cost Estimate, by Area

Process Plant Area	Cost (USD Million)
Primary Crushing	16.018
Reclaim Stockpile	9.265
Grinding	59.346
Flotation and Regrind	63.330
Concentrate Thickening and Filtration	17.180
Tailings Thickening	35.000
Tailings Filtration	32.493
Reagents	12.427
TOTAL	**245.059**

Table 21-7: Process Plant Capital Cost Estimate, by Discipline

Discipline	Cost (USD Million)
Concrete	31.545
Structural	32.002
Architecture	3.979
Mechanical	118.629
Piping	13.274
Electrical	39.833
Instrumentation	5.797
TOTAL	**245.059**

The process plant includes the primary crushing, reclaim stockpile, grinding, flotation and regrind, concentrate thickening and filtration, tailings thickening, tailings filtration and reagents elements of the work breakdown structure.

GMI estimated these costs based on equipment lists and bills of quantities that were developed for earthworks, concrete, steel, piping, electrical and instrumentation disciplines based on Project general arrangement drawings, P&ID drawings and other documentation.

All major plant equipment was reinvestigated for this Phase 1 FEED and new pricing was solicited from qualified vendors for the following equipment:

- Primary Crushing
- Hydraulic Hammer
- SAG, Ball and Regrind Mills
- Thickeners
- Concentrate and Tailings Filters
- Hydrocyclones
- Flotation Cells
- Apron Feeders
- Belt Conveyors
- Lime Slaker System
- Compressors
- Dust Collection
- Agitators
- Cranes
- Flocculant Plant Package
- Reagent Plant Package
- Water Treatment Plant Package
- Filtered Tailings Radial Stacker
- Major Electrical Equipment (Transformers, MCCs, and Switchgear)

Supplier pricing was also received for steel structures, piping and electrical cabling. Where quotes were not requested, historical pricing information from similar projects was used.

Process plant capital costs include:

- All labor required for Project construction and management activities (excluding Engineering and Project Management and Supervision)
- All material and equipment required for construction
- Mechanical, electrical, control, instrumentation, civil works, earthworks and piping installation
- Construction and installation.

21.6 Fresh Water / Process Water Capital Costs

The capital costs associated with the management of fresh water and process water are shown in Table 21-8. These costs include water ponds, water storage, pumping and associated pipework.

Table 21-8: Fresh Water / Process Water Capital Cost Estimate

Cost Type	Cost (USD Million)
Water Pond	14.070
Water Management	14.738
Ancillary Process Plant	12.741
TOTAL	**41.549**

21.7 Power Supply Capital Costs

The capital costs associated with the Project power supply are shown in Table 21-9. It should be noted that the main power transmission line to the Project site will be contracted on a build-own-operate-transfer basis and therefore the associated costs are captured in the Project operating costs.

Table 21-9: Power Supply Capital Cost Estimate

Cost Type	Cost (USD Million)
Camp Power Line	0.895
Electrical Distribution System	2.039
TOTAL	**2.934**

21.8 Ancillary Buildings Capital Costs

In Table 21-10 details of the capital costs of onsite ancillary facilities are shown. These include the site entry control point, laboratory, warehouse, plant maintenance building and camp.

Table 21-10: Ancillary Facilities Capital Cost Estimate

Cost Type	Cost (USD Million)
Entry Control Point and Truck Weigh Station	0.446
Laboratory	2.896
Rescue Platform	0.053
Warehouse	3.223
Nitrate and Ammonium Storage	0.474
Plant Maintenance Building	2.624
Camp (Temporary and Operation)	13.261
TOTAL	**22.977**

21.9 Engineering, Project Management & Supervision Capital Costs

Engineering, Project Management and Supervision costs are shown in Table 21-11. Engineering costs include an allowance for Phase 2 detail engineering, field engineering and technical assistance during pre-commissioning, commissioning and ramp-up.

Table 21-11: Engineering and Project Management Capital Cost Estimate

Cost Type	Cost (USD Million)
Engineering	10.000
Project Management & Supervision	14.980
TOTAL	**24.980**

21.10 Commissioning, Start Up and Vendor Representative Capital Costs

The capital cost estimate includes allowance for vendor construction supervision, vendor pre-commissioning, vendor commissioning, commissioning spare parts, capital spare parts and first fills. These costs are broken down and shown in Table 21-12. An allowance for capital spare parts is included in this cost area.

Table 21-12: Commissioning, Start up and Vendor Representative Capital Cost Estimate

Cost Type	Cost (USD Million)
Vendor Construction Supervision	1.235
Vendor Pre-Commissioning	0.370
Vendor Commissioning	0.370
Commissioning Spare Parts	0.618
Capital Spare Parts	6.792
First Fills	4.503
TOTAL	**13.888**

21.11 Owner's Capital Costs

The current CAPEX includes an estimate for Owner's Costs as shown in Table 21-13. These costs include estimates for Owner's staffing during pre-production, site communications, Owner's camp operating costs, operator training, Owner's commissioning, insurance, environmental compliance, community development, land acquisitions, consultants, legal expenses and further metallurgical testing.

Table 21-13: Owner's Capital Cost Estimate

Cost Type	Cost (USD Million)
Staff Build-up	0.500
Communications	0.775
Camp Costs	1.264
Temp Sanitation	2.450
Offices	0.084
Admin Equipment	0.550
Mine & Plant Shop Equipment & Warehouse Fit-out	0.275
Medical, Security & Safety	0.275
Pre-production Employment & Training	2.800
Owner Management	0.840
Owner Commissioning Team	0.480
Insurance	1.080
Corporate Services	0.250
Environmental Compliance	0.600
Community Development	1.500
ROW & Land Acquisition	0.400
Addition Consultants	0.400
Legal, Permits & Fees	0.500
Detailed Metallurgical Testing	1.000
Additional Owner Costs	16.235
TOTAL	**32.258**

21.12 Operating Cost Summary

Operating costs incurred over the life of mine are presented in Table 21-14.

Table 21-14: Operating Cost Summary

Cost Item	LOM Cost (USD Million)
Mining	802.269
Process Plant	1,406.214
Treatment, Refining, Shipping	851.551
General and Administrative	235.595
Reclamation and Closure	29.389
TOTAL	**3,325.018**
LOM ROM tonnes	**139,072,635**
Average LOM operating cost	**23.91 USD/t**

21.13 Mine Operating Costs

21.13.1 Mine Operating Cost Summary

The mine will be operated by a contractor. Costs associated with the contract mining are presented in Table 21-15.

Table 21-15: Contract Mining Operating Cost Estimate

Cost Item	LOM Cost (USD Million)
Direct Costs	646.649
Indirect Costs	155.620
TOTAL	**802.269**

21.14 Process Plant Operating & Maintenance Costs

The process plant operating costs are summarized by areas of the plant and then by cost elements of labor, power, reagents, grinding media, wear items, maintenance parts and supplies and services. A summary of the processing plant operating costs is shown in Table 21-16.

Table 21-16: Process Plant Operating Cost Estimate

Cost Item	LOM Cost (M USD)	LOM Average Cost (USD/t Ore)
Operating & Maintenance Labor	95.916	0.69
Power	329.647	2.37
Reagents	650.380	4.68
Water Pumping	2.781	0.02
Mobile Equipment	46.339	0.33
Maintenance Parts & Services	143.076	1.03
Tailing Disposal	124.998	0.90
Laboratory	13.078	0.09
TOTAL	**1,406.214**	**10.11**

21.14.1 Process Labor & Fringes

Process labor costs were derived from a staffing plan and based on prevailing annual labor rates in the areas which were provided by Bear Creek Mining. Labor rates and fringe benefits for employees include all applicable social security benefits as well as all applicable payroll taxes. A total of 180 employees will be employed at the process plant with 126 in operations and 54 in maintenance. A summary of the staffing plan and gross annual labor costs are shown in Table 21-17. The details of the staffing plan are shown in Table 21-18. An allocation for personnel training has been included in the labor cost.

Table 21-17: Process Plant Staffing Summary

Department	Number of Personnel	Total Labor (M USD/a)
Mill Operations	126	3.372
Mill Maintenance	54	1.266
Training	-	0.696
TOTAL	**180**	**5.334**

Table 21-18: Staffing Plan

Department	Job Title	Number of Personnel
Management	General Operations Manager	1
	Plant Superintendent	1
	Secretary Clerk	2
	Plant Foreman	4
	Grinding Supervisor	4
	Flotation Supervisor	4
	Tailings Supervisor	4
	Replacement Supervisor	1
Process Plant	Plant Metallurgist	1
	Crushing Loader Operator	4
	Crushing Operator	4
	Crushing Helper	4
	Comminution Operator	4
	Grinding Operator	8
	Pb Flotation Operator	4
	Pb Flotation Shift Laborer	4
	Zn Flotation Operator	4
	Zn Flotation Shift Laborer	4
	Control Room Operator	4
	Reagents Operator	4
	Reagents Helper	4
	Filter/Thickener Operator	4
	Filter/Thickener Helper	4
	Tailings Filter Operator	4
	Tailings Filter Helper	4
	Filtration Loader Operator	4
	Day Shift	8
	Shift Labourer	4
Plant Maintenance	Maintenance Superintendent	1

Department	Job Title	Number of Personnel
	Maintenance Planner	1
	Process Maintenance Shift Foreman	4
Mill Maintenance	Mechanic	10
	Mechanic Helper	10
	Welder	2
	Welder Helper	2
	Electrician	8
	Electrician Helper	8
	Crane Operator	4
	Instrumentation Tech	4
Laboratory and Metallurgy	Chief Metallurgist	1
	Senior Metallurgist	1
	Junior Metallurgist	2
	Metallurgical Helper	3
	Chief Chemist	1
	Lab Technician	6
	Shift Sample Buckers	6
TOTAL		**180**

21.14.2 Power

Power costs were based on purchasing power from a local utility company and associated rates were applied. Power consumption was based on the connected kW derived from the equipment list, discounted for operating time per day and anticipated operating load level. The overall power cost is estimated at $0.0557 per kWh with a consumption of 42.3 kWh per ore tonne. A summary of the power consumption and costs are shown in Table 21-19.

Table 21-19: Summary of Electric Power Consumption and Cost Estimate

Cost Item	Installed Power (kW)	Average 24 Hour Demand (kW)	Average Power (MWh/a)
Area 100 - Crushing	1,149.9	556.0	4,803.5
Area 200 - Crushed Ore Stockpile	358.8	202.3	1,747.8
Area 300 - Grinding	25,553.6	20,474.9	176,903.2
Area 400 - Flotation & Regrinding	12,257.3	7,255.6	62,688.0
Area 500 - Concentrates Thickening and Filtration	2,021.7	1,174.0	10,143.4
Area 600/650 - Tailing Thickening and Filtration	13,358.7	8,400.6	72,580.9
Area 800 - Reagents	810.6	322.4	2,785.8
Area 900 - Services	1,595.6	143.6	1,241.1
TOTAL	**57,106**	**38,529**	**332,894**
USD/kWh			**0.0557**
M USD/a			**18.546**
USD/tonne ore			**2.355**

21.14.3 Reagents

Consumption rates were determined from the metallurgical test data or derived, or estimated from industry practice. Reagents prices were supplied by Bear Creek Mining from local sources in the area with an allowance for freight to site. A summary of process reagent consumption and costs is shown in Table 21-20.

Table 21-20: Summary of Reagents Consumption and Cost Estimate

Reagent	Consumption		Unit Rate	
	kg/t	Average kg/a	USD/kg	Average M USD/a
Sodium Isopropyl Xanthate	0.040	309,090	1.74	0.538
Lime	3.500	27,045,400	0.14	3.786
Methyl Isobutyl Carbinol	0.050	386,363	2.38	0.920
A-404	0.015	115,909	2.55	0.296
Sodium Cyanide	0.210	1,622,724	2.08	3.375
Copper Sulphate	0.300	2,318,177	1.99	4.613
Sodium Hydroxide	0.010	77,273	0.85	0.066
Sodium Bisulphite	0.500	3,863,629	0.66	2.550
Zinc Sulphate	0.620	4,790,899	1.00	4.791
Flocculant	0.020	154,545	3.02	0.467
Antiscalant	0.005	38,636	2.4	0.093
Laboratory Analyses	Calculated from estimated number of samples			0.741
TOTAL				**22.236**

21.14.4 Maintenance, Wear Parts and Consumables

Grinding media consumption and wear items (liners) were based on industry practice for the crusher and grinding operations. These consumption rates and unit prices are shown in Table 21-21.

Table 21-21: Grinding Media and Wear Parts Consumption and Cost Estimate

Item	Consumption		Unit Rate	
	kg/t	Average kg/a	USD/kg	Average M USD/a
Primary Crusher Liners	0.008	61,818	4.6	0.284
SAG Mill Liners	0.050	386,363	3.6	1.391
SAG Mill Balls	0.500	3,863,629	1.16	4.482
Ball Mill Liners	0.030	231,818	3.6	0.835
Ball Mill Balls	0.500	3,863,629	0.97	3.748
Lead Regrind Mill Liners	0.005	38,636	5.6	0.216
Zinc Regrind Mill Liners	0.005	38,636	5.6	0.216
Lead Regrind Mill Balls	0.010	77,273	0.96	0.074
Zinc Regrind Mill Balls	0.010	77,273	0.96	0.074
Maintenance	Calculated as percentage of each area CAPEX			6.028
TOTAL				**17.348**

Allowances were made to cover the cost of maintenance of all items that were not specifically identified and to cover the cost of maintenance of the facilities, calculated as a percentage of CAPEX for each area.

21.14.5 Process Supplies & Services

Allowances were provided in process plant for outside consultants, outside contractors, vehicle maintenance, and miscellaneous supplies. These were provided by the Client, and are presented as part of the G&A cost shown in Table 21-23.

Laboratory

Laboratory cost estimates are based on labor and fringe benefits, power, reagents, assay consumables, and supplies and services. The laboratory and associated labor costs are included in the process plant operating cost calculation summarised in Table 21-15 and Table 21-18.

21.15 Concentrate Handling, Transportation and Storage Operating Costs

Concentrate will be loaded into supersacks and then loaded onto trucks, which will haul the supersacks to the port of Maturani where they will be stored and loaded onto a ship to be taken to the smelter for further processing. Costs associated with concentrate handling and transportation are presented in Table 21-22.

Table 21-22: Concentrate handling, treatment, and refining costs

Cost Item	LOM Cost (USD Million)
Concentrate Treatment Charges	372.358
Silver Refining Charges	137.343
Transportation	341.850
TOTAL	**851.551**

21.16 General Services and Administration (G&A) Operating Costs

The operating cost for the G&A areas were determined by the Client and summarized by cost element as shown in Table 21-23. The cost elements include labor, supplies, support infrastructure, services, and other expenses. In addition to these costs, a 5% contingency was added. The departments that contribute to the G&A costs include administration, human resources, purchasing, safety and environmental.

Table 21-23: G&A Operating Cost Estimate

Cost Item	LOM Cost (M USD)	LOM Cost (USD/t)
Labor and Fringes	42.988	0.309
Power	5.396	0.039
Vehicle Operating and Maintenance	0.546	0.004
Communications	2.727	0.020
Safety Supplies / Incentives	1.800	0.013
Offsite Training and Conferences	0.540	0.004
Insurance	19.332	0.139
Corporate Services and Travel	8.100	0.058
Environmental	1.944	0.014
Security and Medical	3.600	0.026
Professional Membership Costs	0.072	0.001
Community Development	27.000	0.194
Staff Living Expenses	92.493	0.665

Cost Item	LOM Cost (M USD)	LOM Cost (USD/t)
Consultants	0.900	0.006
Computer Equipment / Software	0.630	0.005
Misc. Office Supplies	0.324	0.002
Misc. Freight and Couriers	0.270	0.002
Recruiting and Relocation	3.564	0.026
Mine Access Road Maintenance	7.200	0.052
Legal, Permits, Fees	4.950	0.036
Contingency (5 %)	11.219	0.081
TOTAL	**235.595**	**1.69**

21.17 Reclamation and Closure Cost

Costs associated with the closure and reclamation were determined by the Client, and are summarised in Table 21-24. Costs include progressive closure costs that occur during the life of mine, final closure costs that occur after production has ceased and post closure costs that are associated with ongoing monitoring and maintenance activities. These costs are expected to be partially offset by salvaging some items.

Table 21-24: Closure Cost Estimate

Item	Cost (USD Million)
Reclamation and Closure Cost	36.292
Salvage Cost Savings	(6.903)
TOTAL	**29.389**

22 Economic Analysis

GRE has reviewed and verified the economic model generated by BMC with capital and operating cost inputs from GMI and GyM, and Anddes and that the model was prepared with sound engineering and financial principles and is correct. GRE has found the work performed by GMI/GyM for the Detailed Engineering Phase 1 (FEED) to be well done and thorough. The financial indicators stated herein are slightly improved over the M3/GRE 2015 Feasibility Study Estimate.

The following text describes the work done by BCM, GMI/GyM, and Anddes.

22.1 Introduction

The financial evaluation presents the determination of the net present value (NPV), payback period (time in years to recapture the initial capital investment), and the internal rate of return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures, production cost, and sales revenue. Revenues are based on the production of a zinc concentrate with silver and a lead-silver concentrate. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

22.2 Mine Production Statistics

Mine production is reported as ore and waste from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report. A total of 139.0 million tonnes of ore are mined at an average grade of 50.3 g/t silver, 0.896% lead, and 0.590% zinc. A total of 207.1 million tonnes of waste are mined for a stripping ratio of 1.49:1.

22.3 Plant Production Statistics

The design basis for the process plant is 22,500 tonnes per day at 92% mill availability. The metal recoveries, which are variable by ore characteristics, are projected to average 67.1% for zinc, 61.1% for lead and 69.9% for silver.

The estimated life of mine metal production is presented in Table 22-1.

Table 22-1: Metal Production

	Life of Mine
Zinc (million lbs)	1,213.3
Lead (million lbs)	1,677.0
Silver (million ozs)	157.2

22.4 Smelter and Refinery Return Factors

The lead and zinc concentrates will be transported to a holding facility at the port of Matarani and consolidated for shipment to a smelter for final processing. Smelter treatment and refining charges will be negotiated at the time of the finalization of the sales agreements. The smelter charges used in the financial model are presented in Table 22-2 and Table 22-3.

Table 22-2: Smelter Treatment Factors (Lead Concentrate)

Lead Concentrate	
Payable lead	95.0%
Minimum Deduction (%)	3.0%
Payable silver	95.0%
Ag Minimum Deduction (oz/dmt)	1.61
Treatment charge ($/dmt)	$150.00
Refining charge – Ag ($/payable oz.)	$1.00
Lead Concentrate Transportation	
Concentrate Trucking and port ($/wmt)	$40.00
Concentrate Shipping ($/wmt)	$90.00
Moisture	8.0%
Penalties	
Arsenic $/dmt per 0.10% over 0.2%	$2.00
Tin $/dmt per 0.10% over 0.2%	$1.50
Bismuth $/dmt per 0.01% over 0.1%	$2.00
Mercury – $/dmt per 10 ppm over 50 ppm	$2.00
Zinc $/dmt per 1% over 8%	$1.00

Table 22-3: Smelter Treatment Factors (Zinc Concentrate)

Zinc Concentrate	
Payable zinc	85.0 %
Minimum Deduction (%)	8.0 %
Price Participation Basis ($/tonne metal)	
Price Participation between $2,500 - $3,000 - $/dmt	$0.06
Price Participation between $2,500 - $2,000 – $/dmt	($0.02)
Payable silver (% of balance)	74.0 %
Silver Minimum Deduction (oz/dmt)	3.0
Treatment charge ($/dmt)	$145.00
Refining charge – Ag (% of metal price)	0.0 %
Zinc Concentrate Transportation	
Concentrate Trucking and port ($/wmt)	$47.26
Concentrate Shipping ($/wmt)	$72.36
Moisture	8.0%
Penalties	
Arsenic $/dmt per 0.1% over 0.1%	$2.00
Cadmium $/dmt per 0.1% over 0.4%	$1.00
Iron $/dmt per 1% over 8%	$2.00
Mercury – $/dmt per 10 ppm over 50 ppm	$2.00
Antimony - $/dmt per 0.01% over 0.1%	$1.00
Copper + Lead - $/dmt per 1% over 4%	$2.00
Silica $/dmt per 1% over 0.5%, >3% may be unacceptable	$0.20

22.5 Capital Expenditure

22.5.1 Initial Capital

The base case financial indicators have been determined using the assumption of 100% equity financing of the initial capital. The total initial capital estimate for the project, which includes pre-production mine development, construction, owners' costs and contingency is $585.6 million. A breakout of the capital cost is shown in Section 21. Approximately 68% of these expenditures will be incurred over a two-year period.

22.5.2 Sustaining Capital

Sustaining capital for equipment was included in the Contractor's costs and was estimated to be $0.36 million.

22.5.3 Working Capital

A 15-day delay of revenue recognition until receipt of cash has been used for accounts receivables. A delay of payment for accounts payable of 30 days is also incorporated into the financial model.

The financial model assumes that the project will qualify for early recovery of Peruvian value added taxes (IGV) during the development and construction phase of the project. Therefore, the financial model reflects a 90-day delay between initial payment and subsequent recovery of 18% IGV taxes. All the working capital is recaptured at the end of the mine life, and the final value of these accounts is $0.

22.5.4 Revenue

Annual revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. Revenue is the gross value of payable metals sold before treatment charges and transportation charges. The metal price assumptions used in the economic model are as follows:

- Zinc $1.10/lb
- Lead $0.95/lb
- Silver $18.00/oz

Total revenue for the life of the mine is summarized in Table 22-4.

Table 22-4: Revenue Summary

Revenue	Gross $	Penalties $	Price Participation $	Net Revenue $
Zinc Concentrate	$1,261.3 million	$8.2 million	-$1.5 million	$1,254.6 million
Lead Concentrate	$3,971.4 million	$18.5 million	$0 million	$3,952.9 million
Total	$5,232.7 million	$26.7 million	-$1.5 million	$5,207.5 million

22.5.5 Total Operating Cost

The average Total Operating Cost over the life of the mine is estimated to be $21.50 per tonne of ore processed, excluding the cost of the capitalized pre-stripping. Total Operating Cost includes mine operations, process plant operations, general and administrative cost, smelting and refining charges, and shipping charges. Table 22-5 shows the estimated operating cost by area per tonne of ore processed.

Table 22-5: Life of Mine Operating Cost

Operating Cost	$/ore tonne
Mine	$5.77
Process Plant	$10.11
General Administration & Laboratory	$1.69
Smelting/Refining Treatment & Concentrate Transport	$3.92
Total Operating Cost	$21.50

Note: Total shown in table is inconsistent because of rounding of the inputs

22.6 Total Cash Cost

The average Total Cash Cost over the life of the mine is estimated to be $5.01 per ounce of payable silver, including reclamation and net of lead and zinc credits. Total Cash Cost for the project is summarized in Table 22-6.

Table 22-6: Life of Mine Total Cash Cost

Direct Production Costs	$2,469,300
Transportation, Treatments, and Refining Charges	$851,550
Subtotal	**$3,320,850**
Lead Payable Revenue	($1,499,247)
Zinc Payable Revenue	($1,133,533)
Total Cash Cost, Net of Lead and Zinc Revenues	**$688,050**
Reclamation	$36,292
Total Cash Cost, Including Reclamation	**$724,342**
Payable Silver Ounces	144,439
Total Cash Cost per Ounce of Payable Silver, Net of Lead and Zinc Revenues	**$5.01**

22.6.1 Salvage Value

A $6.9 million allowance for salvage value has been included in the cash flow analysis.

22.6.2 Reclamation & Closure

An allowance of $36.3 million for the cost of reclamation and closure of the property has been included in the cash flow projection.

22.7 Depreciation

Depreciation was calculated using the following assumptions for both initial and sustaining capital.

- Mine capital 5-year straight line method
- Process Plant and Infrastructure 10-year straight line method
- Last year of production is the catch-up year if assets are not fully depreciated

22.8 Taxation

22.8.1 Contribution to OSINERGMIN

The contribution to OSINERGMIN is applied to Net Income after Interest at a rate of 0.13%. It is estimated that $6.8 million of contribution to OSINERGMIN will be paid during the life of the mine.

22.8.2 Contribution to OEFA

The contribution to OEFA is applied to Net Income after Interest at a rate of 0.13%. It is estimated that $5.8 million of contribution to OEFA will be paid during the life of the mine.

22.8.3 Royalty Tax

The royalty tax is applied to operating profit at progressive rates from 1% to 12% based on operating margin (operating profit divided by sales), subject to a minimum tax of 1% of sales, which is applicable regardless of the Company's operating margin. It is estimated that $52.3 million of royalty tax will be paid during the life of the mine.

22.8.4 Special Tax (IEM)

A special tax (IEM) is applied to operating profit at progressive rates from 2% to 8.4% based on operating margin (operating profit divided by sales). It is estimated that $13.3 million of special tax will be paid during the life of the mine.

22.8.5 Worker's Participation Tax

A labor profit sharing tax is generally based on pre-tax profits, after deduction for the royalty tax and special tax (IEM), and is assessed at an 8% rate. It is estimated that $37.3 million of labor profit sharing tax will be paid during the life of the mine.

22.8.6 Income Tax

Income taxes are assessed on pre-tax profits at a rate of 29.5%, after deduction for the special tax (IEM), royalty tax, and worker's participation tax. It is estimated that $126.6 million of income taxes will be paid during the life of the mine.

22.9 Net Income After Tax

Net income after taxes for the project amounts to $863.9 million.

22.10 Project Financing

The financial model has been prepared on the assumption that the project will be financed 100% with equity.

22.11 Net Present Value, Internal Rate of Return, Payback

The economic analyses for the project are summarized in Table 22-7.

Table 22-7:Financial Analysis Results

($millions)	Pre-Tax	After Tax
NPV @ 0% ($000)	$1,301	$887
NPV @ 5% ($000)	$648	$405
IRR	19.4%	15.1%
Payback (Years)	3.0	3.6

22.12 Sensitivity Analysis

The results of the sensitivity analysis for the project both before taxes and after taxes are shown in Table 22-8 to Table 22-11 and Figure 22-1 to Figure 22-4.

Table 22-8: NPV Sensitivity Analysis @ 5% - Before Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	1,267,116	369,660	541,248	1,121,667
10%	957,337	508,609	594,403	884,613
0%	647,559	647,559	647,559	647,559
-10%	337,780	786,508	700,714	410,506
-20%	28,001	925,457	753,869	173,455

Table 22-9: IRR% Sensitivity Analysis – Before Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	28.57%	14.65%	15.57%	27.03%
10%	24.28%	17.17%	17.37%	23.41%
0%	19.43%	19.43%	19.43%	19.43%
-10%	13.64%	21.49%	21.81%	14.94%
-20%	5.82%	23.41%	24.63%	9.66%

Table 22-10: NPV Sensitivity Analysis @ 5% - After Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	817,844	209,437	298,200	719,972
10%	611,913	308,566	351,355	562,745
0%	404,511	404,511	404,511	404,511
-10%	193,156	499,265	457,666	244,402
-20%	(44,472)	593,286	510,821	75,938

Table 22-11: IRR% Sensitivity Analysis – After Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	22.61%	10.88%	11.50%	21.26%
10%	19.07%	13.13%	13.17%	18.31%
0%	15.08%	15.08%	15.08%	15.08%
-10%	10.32%	16.85%	17.31%	11.47%
-20%	3.35%	18.47%	19.95%	7.14%

Figure 22-1: Sensitivity Analysis on NPV (before tax) at 5%

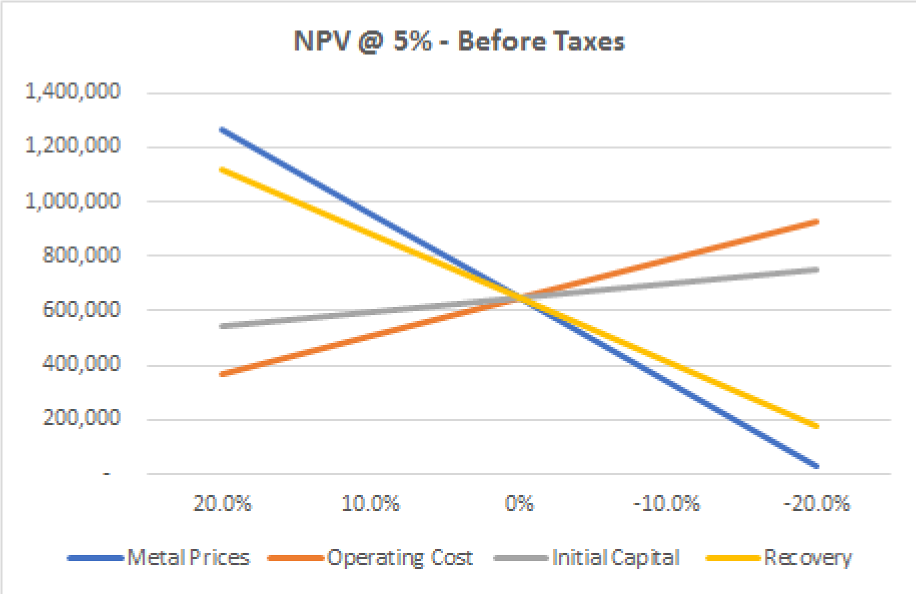


Figure 22-2: Sensitivity Analysis on IRR% (before tax)

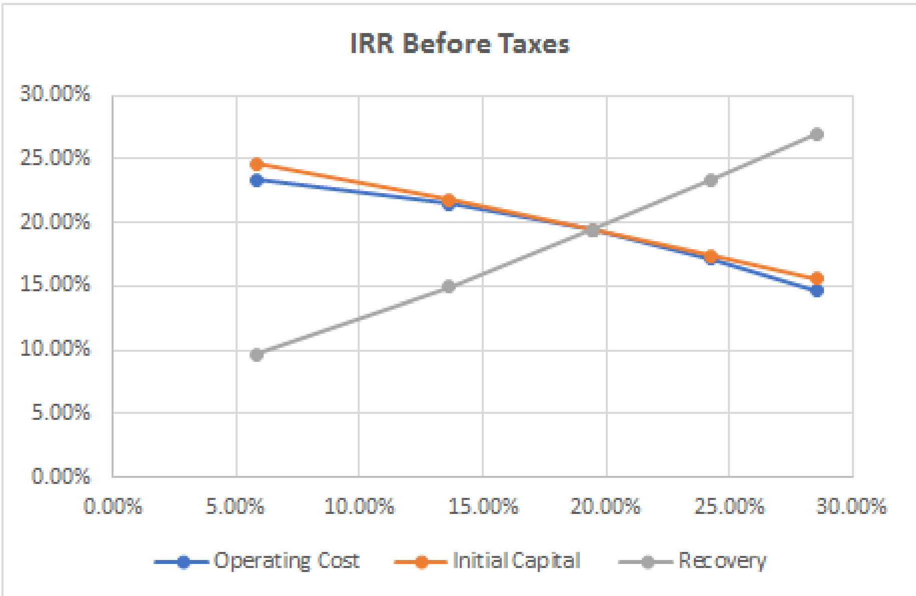


Figure 22-3: Sensitivity Analysis on NPV (after tax) at 5%

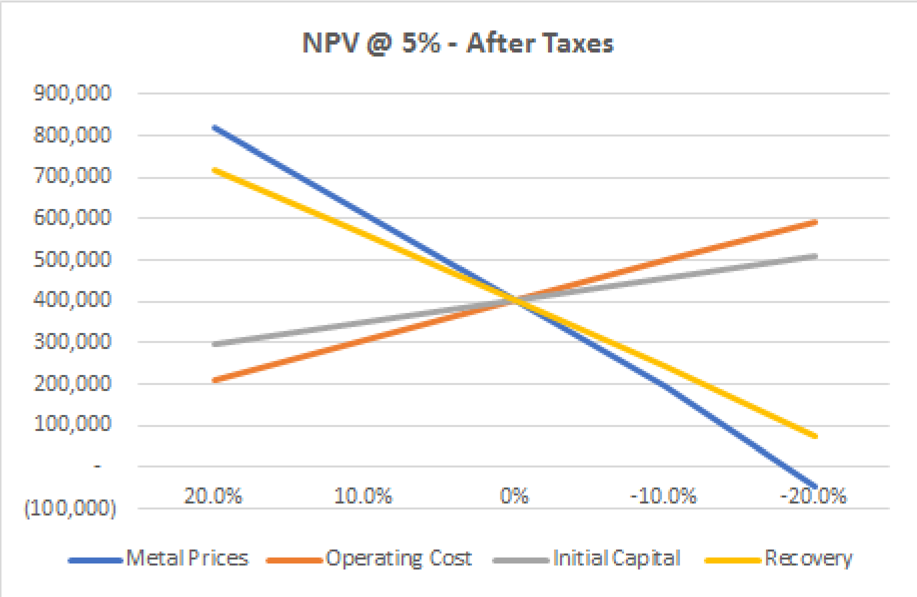
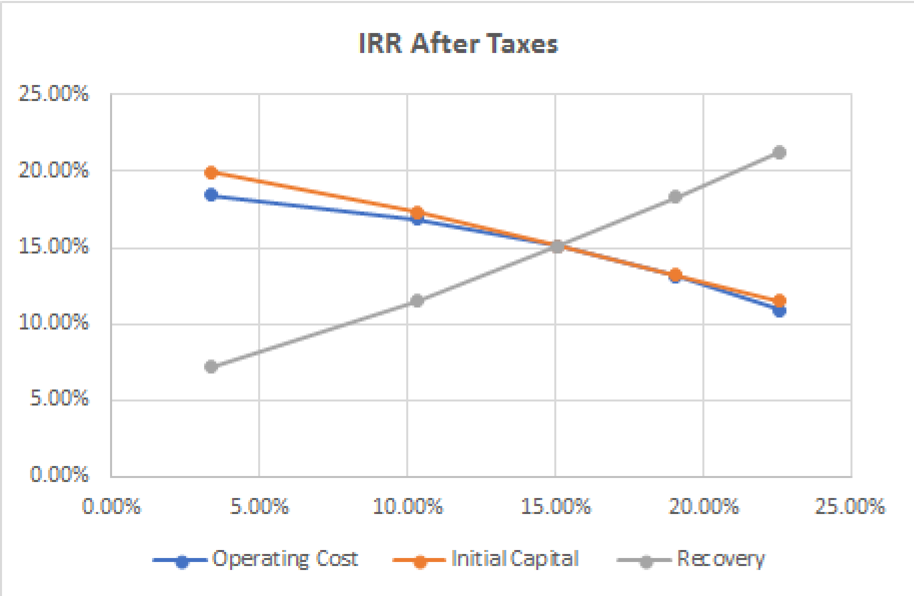


Figure 22-4: Sensitivity on IRR% (after tax)



22.13 Detailed Financial Model

The detailed financial model is presented in Table 22-12.

Table 22-12: Detailed Financial Model

22,500 tpd		Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Mining Operations																											
Ore to Mill																											
Beginning Inventory (kt)	$	139,073	139,073	139,073	139,073	139,073	139,073	139,073	133,744	125,869	117,994	110,119	102,244	94,369	86,494	78,619	70,744	62,869	54,994	47,119	39,244	31,369	23,494	15,619	7,744	-	
Mined (kt)	$	139,073	-	-	-	-	-	5,329	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,744	-	
Ending Inventory (kt)	$	-	139,073	139,073	139,073	139,073	139,073	133,744	125,869	117,994	110,119	102,244	94,369	86,494	78,619	70,744	62,869	54,994	47,119	39,244	31,369	23,494	15,619	7,744	-	-	
Silver Grade (g/t)	$	50	-	-	-	-	-	94.97	77.03	92.82	64.15	60.09	69.13	28.41	38.65	60.00	56.55	40.20	34.89	36.89	37.63	35.96	33.52	31.31	26.74	-	
Zinc Grade (%)		0.59%	0.000%	0.000%	0.000%	0.000%	0.000%	0.738%	0.976%	0.753%	0.694%	0.512%	0.907%	0.730%	0.578%	0.161%	0.233%	0.318%	0.736%	0.588%	0.504%	0.486%	0.451%	0.517%	0.790%	0.000%	
Lead Grade (%)		0.90%	0.000%	0.000%	0.000%	0.000%	0.000%	1.142%	1.061%	1.240%	1.127%	1.076%	1.142%	0.805%	0.704%	0.886%	0.962%	0.811%	0.696%	0.765%	0.748%	0.664%	0.736%	0.896%	0.739%	0.000%	
Contained Silver (kozs)	$	224,778	-	-	-	-	-	16,271	19,503	23,502	16,243	15,213	17,504	7,194	9,785	15,191	14,318	10,177	8,833	9,340	9,527	9,104	8,488	7,928	6,657	-	
Contained Zinc (klbs)	$	1,809,005	-	-	-	-	-	86,732	169,435	130,673	120,481	88,882	157,502	126,712	100,368	27,940	40,383	55,294	127,705	102,151	87,548	84,361	78,288	89,722	134,829	-	
Contained Lead (klbs)	$	2,746,154	-	-	-	-	-	134,162	184,274	215,199	195,595	186,835	198,343	139,700	122,174	153,876	166,973	140,726	120,891	132,881	129,868	115,283	127,702	155,487	126,187	-	
Total Ore																											
Beginning Inventory (kt)	$	139,073	-	-	139,073	139,073	139,073	139,073	133,744	125,869	117,994	110,119	102,244	94,369	86,494	78,619	70,744	62,869	54,994	47,119	39,244	31,369	23,494	15,619	7,744	-	
Mined (kt)	$	139,073	-	-	-	-	-	5,329	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,744	-	
Ending Inventory (kt)	$	-	-	-	139,073	139,073	139,073	133,744	125,869	117,994	110,119	102,244	94,369	86,494	78,619	70,744	62,869	54,994	47,119	39,244	31,369	23,494	15,619	7,744	-	-	
Silver Grade (g/t)	$	50	-	-	-	-	-	94.97	77.03	92.82	64.15	60.09	69.13	28.41	38.65	60.00	56.55	40.20	34.89	36.89	37.63	35.96	33.52	31.31	26.74	-	
Zinc Grade (%)		0.59%	-	-	0.00%	0.00%	0.00%	0.74%	0.98%	0.75%	0.69%	0.51%	0.91%	0.73%	0.58%	0.16%	0.23%	0.32%	0.74%	0.59%	0.50%	0.49%	0.45%	0.52%	0.79%	0.00%	
Lead Grade (%)		0.90%	-	-	0.00%	0.00%	0.00%	1.14%	1.06%	1.24%	1.13%	1.08%	1.14%	0.80%	0.70%	0.89%	0.96%	0.81%	0.70%	0.77%	0.75%	0.66%	0.74%	0.90%	0.74%	0.00%	
Contained Silver (kozs)	$	224,778	-	-	-	-	-	16,271	19,503	23,502	16,243	15,213	17,504	7,194	9,785	15,191	14,318	10,177	8,833	9,340	9,527	9,104	8,488	7,928	6,657	-	
Contained Zinc (klbs)	$	1,809,005	-	-	-	-	-	86,732	169,435	130,673	120,481	88,882	157,502	126,712	100,368	27,940	40,383	55,294	127,705	102,151	87,548	84,361	78,288	89,722	134,829	-	
Contained Lead (klbs)	$	2,746,154	-	-	-	-	-	134,162	184,274	215,199	195,595	186,835	198,343	139,700	122,174	153,876	166,973	140,726	120,891	132,881	129,868	115,283	127,702	155,487	126,187	-	
Waste																											
Beginning Inventory(kt)	$	207,067	-	-	207,067	202,331	202,331	194,756	171,669	148,252	132,864	118,028	104,483	95,362	92,021	86,938	79,643	73,476	59,714	47,101	36,848	25,536	14,099	10,155	6,123		
Mined (kt)	$	207,067	-	-	4,735	-	7,575	23,086	23,417	15,388	14,837	13,544	9,121	3,342	5,082	7,296	6,166	13,763	12,613	10,253	11,312	11,437	3,943	4,032	6,123		
Ending Inventory (kt)	$	-	-	-	202,331	202,331	194,756	171,669	148,252	132,864	118,028	104,483	95,362	92,021	86,938	79,643	73,476	59,714	47,101	36,848	25,536	14,099	10,155	6,123	0		
Total Material Mined (kt)	$	346,139	-	-	4,735	-	7,575	28,415	31,292	23,263	22,712	21,419	16,996	11,217	12,957	15,171	14,041	21,638	20,488	18,128	19,187	19,312	11,818	11,907	13,867		
Waste to Ore Ratio	$	1	-	-	-	-	-	4.33	2.97	1.95	1.88	1.72	1.16	0.42	0.65	0.93	0.78	1.75	1.60	1.30	1.44	1.45	0.50	0.51	0.79		
Process Plant Operations																											
Ore Milled																											
Ore Processed (kt)	$	139,073	-	-	-	-	-	5,329	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,744		
Silver Grade (g/t)	$	50	-	-	-	-	-	94.97	77.03	92.82	64.15	60.09	69.13	28.41	38.65	60.00	56.55	40.20	34.89	36.89	37.63	35.96	33.52	31.31	26.74		
Zinc Grade (%)		0.59%	-	-	-	-	-	0.74%	0.98%	0.75%	0.69%	0.51%	0.91%	0.73%	0.58%	0.16%	0.23%	0.32%	0.74%	0.59%	0.50%	0.49%	0.45%	0.52%	0.79%		
Lead Grade (%)		0.90%	-	-	-	-	-	1.14%	1.06%	1.24%	1.13%	1.08%	1.14%	0.80%	0.70%	0.89%	0.96%	0.81%	0.70%	0.77%	0.75%	0.66%	0.74%	0.90%	0.74%		
Contained Silver (kozs)	$	224,778	-	-	-	-	-	16,271	19,503	23,502	16,243	15,213	17,504	7,194	9,785	15,191	14,318	10,177	8,833	9,340	9,527	9,104	8,488	7,928	6,657		
Contained Zinc (klbs)	$	1,809,005	-	-	-	-	-	86,732	169,435	130,673	120,481	88,882	157,502	126,712	100,368	27,940	40,383	55,294	127,705	102,151	87,548	84,361	78,288	89,722	134,829		
Contained Lead (klbs)	$	2,746,154	-	-	-	-	-	134,162	184,274	215,199	195,595	186,835	198,343	139,700	122,174	153,876	166,973	140,726	120,891	132,881	129,868	115,283	127,702	155,487	126,187		
Zinc Concentrate																											
Ag Recovery (%)		5.63%	-	-	-	-	-	3.88%	6.65%	3.95%	5.33%	4.21%	7.02%	13.51%	7.37%	0.40%	1.22%	2.88%	10.61%	7.03%	6.13%	5.90%	6.13%	7.80%	14.88%		
Zinc Recovery (%)		67.07%	-	-	-	-	-	68.60%	73.02%	66.36%	68.86%	67.51%	73.44%	74.21%	70.36%	20.28%	39.98%	49.26%	69.92%	62.44%	64.58%	62.02%	64.97%	67.64%	72.32%		
Zinc Concentrate (kt)	$	1,022	-	-	-	-	-	51	105	75	70	52	99	79	58	5	14	24	76	53	47	43	42	50	80		
Zinc Concentrate Grade (%)		54%	-	-	-	-	-	52.96%	53.58%	52.49%	53.78%	52.65%	52.84%	54.33%	54.98%	52.00%	52.00%	52.12%	53.48%	54.53%	54.33%	54.69%	54.92%	55.09%	55.28%		
Recovered Silver (kozs)	$	12,656	-	-	-	-	-	631	1,297	927	866	640	1,229	972	721	61	174	293	937	657	584	537	520	619	990		
Recovered Zinc (klbs)	$	1,213,267	-	-	-	-	-	59,500	123,729.227	86,708	82,958	60,006	115,666	94,035	70,615	5,666	16,147	27,236	89,285	63,787	56,540	52,323	50,865	60,691	97,508		
Lead Concentrate																											
Ag Recovery (%)		64%	-	-	-	-	-	65.59%	65.37%	67.67%	68.97%	67.93%	66.55%	61.83%	59.65%	56.78%	57.38%	63.51%	60.25%	62.96%	62.88%	66.52%	67.01%	67.26%	63.09%		
Lead Recovery (%)		61%	-	-	-	-	-	59.81%	66.30%	62.43%	68.93%	65.65%	70.66%	74.31%	51.93%	28.29%	33.47%	50.55%	65.71%	62.41%	60.71%	63.05%	67.76%	69.77%	75.95%		
Lead Concentrate (kt)	$	1,494	-	-	-	-	-	72.798	111.114	113.191	111.761	105.267	127.077	94.166	57.552	39.640	50.614	64.067	70.397	74.953	71.569	65.962	78.500	98.411	86.942		
Lead Concentrate Grade (%)		51%	-	-	-	-	-	50.00%	49.88%	53.84%	54.72%	52.85%	50.03%	50.01%	50.00%	49.81%	50.08%	50.36%	51.18%	50.19%	49.9684%	49.99%	50.00%	50.00%	50.00%		
Recovered Silver (kozs)	$	144,572	-	-	-	-	-	10,673	12,749	15,904	11,203	10,335	11,649	4,448	5,837	8,625	8,216	6,464	5,322	5,881	5,990	6,056	5,687	5,332	4,200		
Recovered Lead (klbs)	$	1,676,964	-	-	-	-	-	80,243	122,182	134,351	134,832	122,657	140,152	103,812	63,440	43,527	55,886	71,136	79,438	82,929	78,842	72,689	86,531	108,479	95,838		
Total Zinc Concentrate																											
Zinc Concentrate (kt)	$	1,022	-	-	-	-	-	50.96266	104.74490	74.9294	69.9749	51.6938	99.2821	79	58	5	14	24	76	53	47	43	42	50	80		
Zinc Concentrate Grade (%)		54%	-	-	-	-	-	52.95832%	53.58037%	52.48989%	53.77512%	52.65257%	52.84435%	54.33%	54.98%	52.00%	52.00%	52.12%	53.48%	54.53%	54.33%	54.69%	54.92%	55.09%	55.28%		
Recovered Silver (kozs)	$	12,656	-	-	-	-	-	631	1,297	927	866	640	1,229	972	721	61	174	293	937	657	584	537	520	619	990		
Recovered Zinc (klbs)	$	1,213,267	-	-	-	-	-	59,500	123,729.227	86,708	82,958	60,006	115,666	94,035	70,615	5,666	16,147	27,236	89,285	63,787	56,540	52,323	50,865	60,691	97,508		
Total Lead Concentrate																											
Lead Concentrate (kt)	$	1,494	-	-	-	-	-	73	111	113	112	105	127.077	94	58	40	51	64	70	75	72	66	78	98	87	-	
Lead Concentrate Grade (%)		51%	-	-	-	-	-	50.00%	49.88%	53.84%	54.72%	52.85%	50.03%	50.01%	50.00%	49.81%	50.08%	50.36%	51.18%	50.19%	49.97%	49.99%	50.00%	50.00%	50.00%	0.00%	
Recovered Silver (kozs)	$	144,572	-	-	-	-	-	10,673	12,749	15,904	11,203	10,335	11,649	4,448	5,837	8,625	8,216	6,464	5,322	5,881	5,990	6,056	5,687	5,332	4,200	-	
Recovered Lead (klbs)	$	1,676,964	-	-	-	-	-	80,243	122,182	134,351	134,832	122,657	140,152	103,812	63,440	43,527	55,886	71,136	79,438	82,929	78,842	72,689	86,531	108,479	95,838	-	
Payable Metals																											
Zinc Concentrate																											
Payable Silver (kozs)	$	7,096	-	-	-	-	-	354	727	520	486	359	689	545	404	34	98	164	526	368	328	301	292	347	555		
Payable Zinc (klbs)	$	1,030,503	-	-	-	-	-	50,512	105,162	73,493	70,483	50,887	98,150	79,930	60,023	4,794	13,663	23,056	75,892	54,219	48,059	44,475	43,235	51,588	82,882		
Lead Concentrates																											
Payable Silver (kozs)	$	137,343	-	-	-	-	-	10,139	12,112	15,109	10,643	9,818	11,067	4,225	5,545	8,194	7,805	6,141	5,056	5,587	5,691	5,753	5,403	5,066	3,990		
Payable Lead (klbs)	$	1,578,154	-	-	-	-	-	75,428	114,833	126,864	127,440	115,695	131,747	97,584	59,633	40,906	52,538	66,899	74,782	77,972	74,108	68,327	81,339	101,971	90,087		
Zinc ($/lb.)	$	1.10	$ -	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10		
Lead ($/lb)	$	0.95	$ -	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95		
Silver ($/oz)	$	18.00	$ -	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00		
Revenues																											
Zinc Concentrates - Zn	$	1,133,553	$ -	$ -	$ -	$ -	$ -	$ 55,563	$ 115,678	$ 80,843	$ 77,532	$ 55,976	$ 107,965	$ 87,922	$ 66,025	$ 5,274	$ 15,029	$ 25,361	$ 83,482	$ 59,641	$ 52,865	$ 48,922	$ 47,559	$ 56,746	$ 91,170		
Zinc Concentrates - Ag	$	127,724	$ -	$ -	$ -	$ -	$ -	$ 6,366	$ 13,084	$ 9,360	$ 8,741	$ 6,457	$ 12,402	$ 9,808	$ 7,277	$ 617	$ 1,759	$ 2,961	$ 9,459	$ 6,628	$ 5,897	$ 5,421	$ 5,248	$ 6,242	$ 9,995		
Lead Concentrates - Pb	$	1,499,247	$ -	$ -	$ -	$ -	$ -	$ 71,657	$ 109,092	$ 120,521	$ 121,068	$ 109,910	$ 125,160	$ 92,704	$ 56,652	$ 38,860	$ 49,911	$ 63,554	$ 71,043	$ 74,073	$ 70,403	$ 64,910	$ 77,272	$ 96,872	$ 85,583		
Lead Concentrates - Ag	$	2,472,174	$ -	$ -	$ -	$ -	$ -	$ 182,502	$ 218,012	$ 271,965	$ 191,579	$ 176,724	$ 199,201	$ 76,055	$ 99,810	$ 147,490	$ 140,497	$ 110,532	$ 91,001	$ 100,563	$ 102,437	$ 103,554	$ 97,255	$ 91,180	$ 71,815		
Total Revenues	$	5,232,698	$ -	$ -	$ -	$ -	$ -	$ 316,089	$ 455,867	$ 482,689	$ 398,920	$ 349,067	$ 444,728	$ 266,490	$ 229,764	$ 192,242	$ 207,197	$ 202,408	$ 254,985	$ 240,905	$ 231,602	$ 222,808	$ 227,335	$ 251,041	$ 258,562		

Detailed Financial Model Continued

22,500 tpd	Total		-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Operating Cost																											
Mining	$ 802,269		$ -	$ -	$ -	$ -	$ -	$ 60,965	$ 71,498	$ 64,816	$ 55,058	$ 51,232	$ 39,777	$ 30,733	$ 35,550	$ 35,779	$ 33,229	$ 42,549	$ 43,756	$ 42,954	$ 44,490	$ 43,843	$ 32,178	$ 33,277	$ 40,587		
Process Plant	$ 1,406,214		$ -	$ -	$ -	$ -	$ -	$ 60,140	$ 80,296	$ 80,296	$ 80,296	$ 80,296	$ 80,296	$ 80,296	$ 80,296	$ 80,296	$ 80,296	$ 80,296	$ 80,296	$ 77,335	$ 77,335	$ 77,335	$ 77,335	$ 77,335	$ 76,148		
General Administration	$ 235,595		$ -	$ -	$ -	$ -	$ -	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089	$ 13,089		
Treatment & Refining Charges																											
Zinc Concentrates																											
Treatment Charges	$ 148,261		$ -	$ -	$ -	$ -	$ -	$ 7,390	$ 15,188	$ 10,865	$ 10,146	$ 7,496	$ 14,396	$ 11,385	$ 8,447	$ 717	$ 2,042	$ 3,437	$ 10,980	$ 7,694	$ 6,845	$ 6,293	$ 6,092	$ 7,246	$ 11,602		
Price Participation	$ (1,532)		$ -	$ -	$ -	$ -	$ -	$ (76)	$ (157)	$ (112)	$ (105)	$ (77)	$ (149)	$ (118)	$ (87)	$ (7)	$ (21)	$ (36)	$ (113)	$ (80)	$ (71)	$ (65)	$ (63)	$ (75)	$ (120)		
Penalty	$ 8,204		$ -	$ -	$ -	$ -	$ -	$ 530	$ 1,089	$ 590	$ 551	$ 407	$ 1,033	$ 487	$ 361	$ 31	$ 87	$ 176	$ 562	$ 394	$ 350	$ 322	$ 312	$ 331	$ 592		
Transportation	$ 132,095		$ -	$ -	$ -	$ -	$ -	$ 6,584	$ 13,532	$ 9,680	$ 9,040	$ 6,678	$ 12,826	$ 10,143	$ 7,526	$ 638	$ 1,820	$ 3,062	$ 9,783	$ 6,855	$ 6,099	$ 5,607	$ 5,428	$ 6,456	$ 10,337		
Lead Concentrate																											
Treatment Charges	$ 224,097		$ -	$ -	$ -	$ -	$ -	$ 10,920	$ 16,667	$ 16,979	$ 16,764	$ 15,790	$ 19,062	$ 14,125	$ 8,633	$ 5,946	$ 7,592	$ 9,610	$ 10,559	$ 11,243	$ 10,735	$ 9,894	$ 11,775	$ 14,762	$ 13,041		
Price Participation	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		
Penalty	$ 18,549		$ -	$ -	$ -	$ -	$ -	$ 2,002	$ 3,056	$ 1,336	$ 1,319	$ 1,242	$ 2,682	$ 395	$ 242	$ 416	$ 531	$ 474	$ 521	$ 315	$ 751	$ 346	$ -	$ 964	$ 1,956		
Silver Refining Charges	$ 137,343		$ -	$ -	$ -	$ -	$ -	$ 10,139	$ 12,112	$ 15,109	$ 10,643	$ 9,818	$ 11,067	$ 4,225	$ 5,545	$ 8,194	$ 7,805	$ 6,141	$ 5,056	$ 5,587	$ 5,691	$ 5,753	$ 5,403	$ 5,066	$ 3,990		
Transportation	$ 209,755		$ -	$ -	$ -	$ -	$ -	$ 10,221	$ 15,600	$ 15,892	$ 15,691	$ 14,779	$ 17,842	$ 13,221	$ 8,080	$ 5,565	$ 7,106	$ 8,995	$ 9,884	$ 10,523	$ 10,048	$ 9,261	$ 11,021	$ 13,817	$ 12,207		
Total Operating Cost	$ 3,320,850		$ -	$ -	$ -	$ -	$ -	$ 181,902	$ 241,970	$ 228,538	$ 212,492	$ 200,750	$ 211,919	$ 177,981	$ 167,682	$ 150,663	$ 153,576	$ 167,793	$ 184,372	$ 175,909	$ 175,363	$ 171,678	$ 162,569	$ 172,267	$ 183,428		
Salvage Value	$ (6,903)		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (6,903)	$ -	
Reclamation & Closure	$ 5,755		$ -	$ -	$ -	$ -	$ -	$ 760	$ 1,130	$ 673	$ 433	$ 216	$ 631	$ 13	$ -	$ 172	$ 184	$ 323	$ 58	$ 1,149	$ 13	$ -	$ -	$ -	$ -		
Reclamation & Closure last 6 years US$ 30,537	$ 30,537		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,896	$ 3,896	$ 3,896	$ 3,896	$ 2,515	$ 505	$ 505	$ 505	$ 505	$ 485	$ 9,935		
Total Production Cost	$ 3,350,239		$ -	$ -	$ -	$ -	$ -	$ 182,662	$ 243,100	$ 229,211	$ 212,925	$ 200,966	$ 212,550	$ 177,994	$ 171,578	$ 154,731	$ 157,656	$ 172,012	$ 186,945	$ 177,562	$ 175,880	$ 172,183	$ 163,074	$ 172,752	$ 186,461		
Operating Income	$ 1,882,459		$ -	$ -	$ -	$ -	$ -	$ 133,426	$ 212,767	$ 253,478	$ 185,995	$ 148,102	$ 232,178	$ 88,496	$ 58,186	$ 37,511	$ 49,541	$ 30,396	$ 68,040	$ 63,343	$ 55,722	$ 50,625	$ 64,261	$ 78,289	$ 72,102		
Corani Amortization	$ 92,482		$ -	$ -	$ -	$ -	$ -	$ 21,697	$ 21,697	$ 21,697	$ 3,913	$ 3,913	$ 3,913	$ 3,913	$ 3,913	$ 3,913	$ 3,913	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		
Depreciation	$ 511,693		$ -	$ -	$ -	$ -	$ -	$ 51,205	$ 51,205	$ 51,205	$ 51,205	$ 51,205	$ 51,133	$ 51,133	$ 51,133	$ 51,133	$ 51,133	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		
Total Depreciation	$ 604,174		$ -	$ -	$ -	$ -	$ -	$ 72,902	$ 72,902	$ 72,902	$ 55,118	$ 55,118	$ 55,046	$ 55,046	$ 55,046	$ 55,046	$ 55,046	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		
Net Income After Depreciation & Amortization	$ 1,278,284		$ -	$ -	$ -	$ -	$ -	$ 60,524	$ 139,865	$ 180,576	$ 130,877	$ 92,983	$ 177,131	$ 33,450	$ 3,140	$ (17,535)	$ (5,506)	$ 30,396	$ 68,040	$ 63,343	$ 55,722	$ 50,625	$ 64,261	$ 78,289	$ 72,102	$ -	
Interest Expense	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		
Net Income After Interest	$ 1,278,284		$ -	$ -	$ -	$ -	$ -	$ 60,524	$ 139,865	$ 180,576	$ 130,877	$ 92,983	$ 177,131	$ 33,450	$ 3,140	$ (17,535)	$ (5,506)	$ 30,396	$ 68,040	$ 63,343	$ 55,722	$ 50,625	$ 64,261	$ 78,289	$ 72,102	$ -	
Taxes	$ 414,383		$ -	$ -	$ -	$ -	$ -	$ 8,660	$ 12,141	$ 60,066	$ 50,809	$ 45,734	$ 29,340	$ 55,310	$ (160)	$ 5,439	$ 1,748	$ 3,077	$ 5,983	$ 11,595	$ 15,418	$ 19,758	$ 21,375	$ 26,028	$ 27,306	$ 14,756	
Total Taxes	$ 414,383		-	$ -	$ -	$ -	$ -	$ 8,660	$ 12,141	$ 60,066	$ 50,809	$ 45,734	$ 29,340	$ 55,310	$ (160)	$ 5,439	$ 1,748	$ 3,077	$ 5,983	$ 11,595	$ 15,418	$ 19,758	$ 21,375	$ 26,028	$ 27,306	$ 14,756	
Net Income After Taxes	$ 863,902		-	-	-	-	-	51,865	127,724	120,509	80,067	47,250	147,792	(21,860)	3,300	(22,974)	(7,253)	27,319	62,057	51,747	40,304	30,867	42,886	52,261	44,796	(14,756)	
			$ -	$ -																							
Cash Flow			$ -	$ -																							
Operating Income after Depreciation & Interest	$ 1,278,284		$ -	$ -	$ -	$ -	$ -	$ 60,524	$ 139,865	$ 180,576	$ 130,877	$ 92,983	$ 177,131	$ 33,450	$ 3,140	$ (17,535)	$ (5,506)	$ 30,396	$ 68,040	$ 63,343	$ 55,722	$ 50,625	$ 64,261	$ 78,289	$ 72,102	$ -	
Add back Depreciation	$ 604,174		$ -	$ -	$ -	$ -	$ -	$ 72,902	$ 72,902	$ 72,902	$ 55,118	$ 55,118	$ 55,046	$ 55,046	$ 55,046	$ 55,046	$ 55,046	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	
Working Capital																											
Account Recievable (90% Cash and 10% 60 d	$ -		$ -	$ -	$ -	$ -	$ -	$ (11,691)	$ (9,067)	$ (2,715)	$ 2,768	$ 2,877	$ (2,924)	$ 5,413	$ 3,556	$ 1,841	$ (91)	$ (7)	$ (1,886)	$ (127)	$ 518	$ 440	$ (59)	$ (933)	$ (570)	$ 9,471	$ 3,188
Accounts Payable (50% Cash 50% 30 days)	$ -		$ -	$ -	$ -	$ -	$ -	$ 7,507	$ (5,023)	$ (3,055)	$ (99)	$ 178	$ 968	$ (1,896)	$ 1,156	$ (429)	$ 813	$ 470	$ 24	$ (999)	$ 316	$ (83)	$ (222)	$ 772	$ 166	$ (8,226)	$ 7,663
IGV Refund	$ 0		$ -	$ (787)	$ (2,018)	$ 174	$ 864	$ 1,767	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	
Total Working Capital	$ -		$ -	$ (787)	$ (2,018)	$ 174	$ 864	$ (2,418)	$ (14,090)	$ (5,770)	$ 2,669	$ 3,054	$ (1,956)	$ 3,517	$ 4,712	$ 1,412	$ 722	$ 462	$ (1,862)	$ (1,127)	$ 834	$ 357	$ (281)	$ (161)	$ (405)	$ 1,244	$ 10,851
Debt Financing	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Expenditures																											
Initial Capital																											
Mine	$ 44,309		$ -	$ -	$ 10,293	$ 9,890	$ 24,126	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ 520,375		$ -	$ 14,458	$ 185,040	$ 212,017	$ 100,157	$ 8,703	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Owners Cost	$ 20,515	$ 170,445	$ 1,865	$ 4,370	$ 5,160	$ 5,160	$ 3,960	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Initial Capital	$ 585,198	$ 170,445	$ 1,865	$ 18,828	$ 200,493	$ 227,066	$ 128,243	$ 8,703	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital																											
Mining	$ 361		$ -	$ -	$ -	$ -	$ -	$ 361	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Sustaining Capital	$ 361		$ -	$ -	$ -	$ -	$ -	$ 361	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Capital Expenditures	$ 585,559	$ 170,445	$ 1,865	$ 18,828	$ 200,493	$ 227,066	$ 128,243	$ 9,064	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sunk Capital	$ (4,279)		$ (1,865)	$ (2,414)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Capital Expenditures	$ 581,279		$ -	$ 16,414	$ 200,493	$ 227,066	$ 128,243	$ 9,064	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Cash Flow before Taxes	$ 1,301,179	$ -	$ -	$ (17,201)	$ (202,510)	$ (226,892)	$ (127,379)	$ 121,945	$ 198,677	$ 247,708	$ 188,664	$ 151,156	$ 230,222	$ 92,013	$ 62,899	$ 38,923	$ 50,263	$ 30,859	$ 66,178	$ 62,216	$ 56,556	$ 50,982	$ 63,979	$ 78,129	$ 71,697	$ 1,244	$ 10,851
Cummulative Cash Flow before Taxes			$ -	$ (17,201)	$ (219,711)	$ (446,604)	$ (573,982)	$ (452,037)	$ (253,360)	$ (5,652)	$ 183,012	$ 334,168	$ 564,390	$ 656,403	$ 719,302	$ 758,225	$ 808,488	$ 839,347	$ 905,525	$ 967,741	$ 1,024,297	$ 1,075,279	$ 1,139,259	$ 1,217,387	$ 1,289,084	$ 1,290,329	$ 1,301,179
								1.0	1.0	1.0	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Taxes																											
Income Taxes	$ 414,383		$ -	$ -	$ -	$ -	$ -	$ 8,660	$ 12,141	$ 60,066	$ 50,809	$ 45,734	$ 29,340	$ 55,310	$ (160)	$ 5,439	$ 1,748	$ 3,077	$ 5,983	$ 11,595	$ 15,418	$ 19,758	$ 21,375	$ 26,028	$ 27,306	$ 14,756	$ -
Cash Flow after Taxes	$ 886,797		$ -	$ (17,201)	$ (202,510)	$ (226,892)	$ (127,379)	$ 113,285	$ 186,536	$ 187,641	$ 137,855	$ 105,423	$ 200,882	$ 36,703	$ 63,059	$ 33,485	$ 48,515	$ 27,782	$ 60,195	$ 50,621	$ 41,138	$ 31,224	$ 42,604	$ 52,100	$ 44,391	$ (13,511)	$ 10,851
Cummulative Cash Flow after Taxes			$ -	$ (17,201)	$ (219,711)	$ (446,604)	$ (573,982)	$ (460,697)	$ (274,160)	$ (86,519)	$ 51,336	$ 156,759	$ 357,641	$ 394,343	$ 457,402	$ 490,887	$ 539,402	$ 567,184	$ 627,378	$ 677,999	$ 719,137	$ 750,361	$ 792,966	$ 845,066	$ 889,457	$ 875,946	$ 886,797
								1.0	1.0	1.0	0.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	

Economic Indicators before Taxes					
NPV @ 0%	0%	1,301,179			
NPV @ 5%	5%	647,559			
IRR %		19.4%			
Payback	Years	3.0			
Economic Indicators after Taxes					
NPV @ 0%	0%	886,797	$	(0) $	0
NPV @ 5%	5%	404,511			
IRR %		15.07%			
Payback	Years	3.63			

23 Adjacent Properties

There are no adjacent mineral properties which might materially affect the interpretation or evaluation of the mineralization or exploration targets of the Corani Project.

24 Other Relevant Data and Information

24.1 Project Execution

GRE has reviewed the Project Execution Plan (PEP) and project risk assessment called a Hazard and Operability study (HAZOP) created for the Corani project by GMI and has summarized the comprehensive documents here. The PEP and HAZOP have been prepared in a professional and industry-standard manner. These plans will form the basis for developing the project on time and on budget, in a safe and efficient manner.

Bear Creek Mining Corporation is committed to carrying out its exploration and development activities of minerals in a professional manner, in accordance with internationally agreed guidelines and principles and to be recognized for Sustainable Development and Corporate Social Responsibility. The participation of community and environmental stewardship are cornerstones of this commitment, to promote a safe and healthy work environment. The Company also has the responsibility to promote its projects for the benefit of shareholders, to generate and sustain future growth, and to contribute to economic prosperity and social progress of the host nation of Peru.

24.1.1 Objectives

The objective of the project is to effectively consolidate the procurement and subcontracting strategy and to carry out the project construction, installation, and assembly in a safe way, leading to the commissioning of the Corani Mining project. The goal is to achieve specified quality and performance standards while maintaining schedule and budgeted costs. A brief summary of the main objectives follow:

- Safety objectives: One of the main objectives is to ensure the safety and security of all persons involved with the project and the environment.
- Community relations objectives: To achieve a harmonious social, economic, and cultural environment with neighboring communities.
- Engineering and Technology Objectives: Ensure that EPCs and their subcontractors use only proven technology and designs. Ensure contractor designs and equipment are consistent with all project standards.
- Quality Work Objectives: Ensure that adequate quality assurance and quality control procedures are effectively used and monitored.
- Cost and schedule objectives: Completing the project on time and below budget.
- Change Management Objectives: Create a BCM designated Project Group to direct and manage CM and EPC companies, and install and manage a system to document and manage variations in volumes and quantities, supported by versions of Material Take Off (MTOs), changes in prices, new unit prices, new scope for the project in accordance with the Capital Expenditure Policy of BCM.
- Procurement and purchase objectives: Ensure that the contracting and purchasing strategy is established and implemented to the maximum practical extent, restricting contracts and packages to a manageable number without increasing the total costs of the project,
- Risk Management Objectives: Assess all risks in all aspects of the project and develop appropriate mitigation strategies for those risks.
- Capitalization Objectives: The Owner Team (OT) of BCM, the CM, and the EPCs, should develop a cost control strategy and a cost control system that will allow the project to be delivered below budget, where each item to be capitalized shall contain distributed indirect costs of the contractor. The system should manage all project costs, invoices, contracts, purchase orders, "as built" plans, photographic report, and lists of all expenses incurred, so the BCM finance group can manage the BCM asset books and initiate the depreciation of the delivered facilities.:

24.1.2 Scope of Project Execution Plan

The 170 page PEP document includes the following main topics:

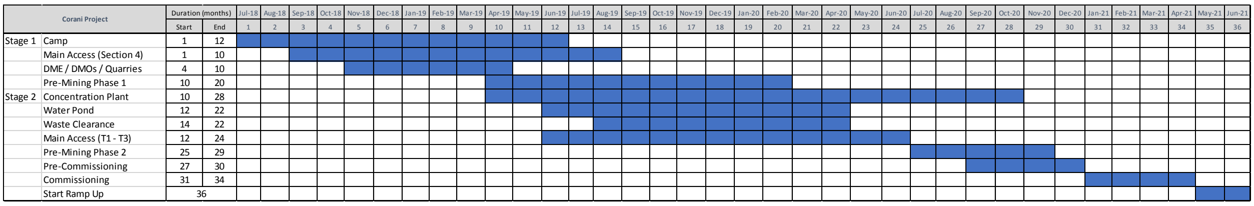
- Background and General Information
- Project Planning and Control
- Execution Strategy and Contractual Packages
- Capex Investment Budget
- Permits Management
- Social Management and Community Relations
- Legal Management
- Execution of Engineering
- Execution of Purchases and Logistics
- Execution of the Construction
- Pre-Commissioning, Commissioning, Start-Up and Ramp Up
- Safety Plan
- Quality Plan
- Environmental Plan

Each section describes the required planning, personnel, companies, systems, controls, work break down structures, procedures, permits, and schedules that need to be developed to efficiently and cost effectively develop the Corani Project.

24.2 Project Development Schedule

The project development continues with engineering and permitting through the remainder of 2017, followed by road construction and long lead equipment procurement beginning the first quarter of 2018. Construction activities continue through 2019 and 2020, with planned commissioning and startup the first quarter of 2021. The schedule shown in Figure 24-1 in preliminary and subject to change.

Figure 24-1: Tentative Project Schedule



24.3 Hazard and Operability Study

Bear Creek Mining Corporation, in conjunction with GMI, have performed a project Hazard and Operability Study or HAZOP and documented that work in the project HAZOP report. The report contains the following sections:

- Background
- Objective of the Shop
- Methodology
- Consequence
- Probability

- Magnitude of Risk
- Shop Restrictions
- Assistants
- Sessions
- Plans
- List of Insepction Areas and Deviations
- Development of Hazop
- Recommendations
- Detail of Inspection Areas

The HAZOP study is designed to identify operational risks as well as potential operational reliability issues, and then develop mitigation plans to guarantee these will not affect or influence the Safety and Environment of Bear Creek Corani Operations. The goal is to have a risk-controlled facility able to meet international safety, security, and environmental standards.

The study has three basic steps: (1) identification of potential operational risk and reliability issues or aspects, (2) determining the consequence of that issue or aspect, including identifing all entities effected, and (3) determining the probability of that issue or aspect occurring and ranking the magnitude of such an occurrence. The HAZOP study includes a method of combining the ranking the consequence and issues or aspects in order probability, to rank the most significant redesign issues or to form mitigation plans for them. The HAZOP study considered only the main or critical circuit; another HAZOP study should be performed at engineering Phase 2 level of development.

The study identified 313 risk points, with 20 of the points being high risk, 105 moderate risk, and 188 low risk. The Study has 53 recommendations, as shown below.

1. Generate a common procedure of operations for the Mine and Plant to control haul truck and vehicle traffic and their destinations.
2. Review the free surface for the stockpile according to the mining plan, in addition to the ore management philosophy; the design includes an area to stockpile ore.
3. Maintenance of ditches and rainwater collection system to ensure proper function throughout the mine life.
4. The maintenance manual should indicate the need for maintenance and cleaning of the area. The parameter of analysis was the frequency of arrival of trucks from the mine, in which the risk is that the trucks at the point of unloading result in an accident, loss of truck, loss of production, etc. In general cases, berms should be used at the platform perimeters and in the unloading point. Safe procedures will always be utilized, and training should be included in the operations manual.
5. The plan will confirm the need for a roof over the hopper during normal precipitation, and, in case of torrential rains, the crushing operation must be stopped, having as backup load live in the stockpile.
6. Review the seepage design; the design contemplates waterproofing of the platform.
7. Review the dust mitigation design to avoid the lack of visibility in the area; the design should contemplate installation of nebulizers to mitigate dust in the environment. Based on a specialist evaluation, the project has contemplated the installation of a nebulizer in the area.
8. The available spaces must be secured for adequate maintainability; the mechanical arrangement of crushing has been designed considering the spaces for the mentioned activities, as well as facilities for lifting.
9. To avoid accumulation of ore, the discharge chute has been dimensioned at 100%.
10. For power outages, the belts and pan feeders must be designed to start with load. The plan will review the need for an estimated time needed for an alternate source of in case of a blackout.

11. Review spare parts and corrective maintenance activities for belt conveyors, as well as for pan feeders; the operations manual shall contain the spare parts recommendations for corrective and preventive maintenance.
12. Perform preventive maintenance of water pump under pressure (water jet).
13. Perform plant start-up according to operations
14. For the SAG mill, review recommendations by suppliers on spaces and services required; the design contemplated the requirement and the recommendations of the suppliers.
15. Generate protocols and maintenance manuals for plant equipment, in this specific case of the trommel and SAG mill. The operations manual should contain recommendations for the maintenance of the plant equipment. Based on this information, the operator will prepare its maintenance plan and procedures, instructions, inspection records to be used in maintenance activities.
16. The plan will review the location and protection of the lubrication system. It will ensure that that the indications and recommendations of the seller of the SAG mill with respect to the location and protection of the lubrication are properly considered.
17. Review with supplier the need for lubrication system protection.
18. Illumination is being considered with level of required luxes indicated along with the project design criteria; the design should contemplate localized lighting.
19. The risk for damage to the bearings due to lack of lubrication in the case of an electrical failure. An available control to consider is an oil lubrication design which does not lose pressure. The recommendation is to review the maintenance and cleaning plan and check the backup system for powering the equipment auxiliary / mill services.
20. Ensure the design prevents spills and allows access for cleaning from accidental events.
21. For the pump boxes for the mills and in general for all pump boxes, the design should consider level controls to prevent spills.
22. In general for pump boxes, evaluate the use of radar sensors.
23. Drainage lines and cleaning lines have been considered for pump suction and discharge pipes, as well as gutters for the concrete slabs.
24. In general for the whole plant, consider maintenance access; the design has contemplated the spaces and access for maintenance.
25. For the feeding of cyclone, the operation conditions and instrumentation for the measurement of solids should be considered.
26. Include a review of the operation of the agitator in the commissioning and maintenance procedures.
27. Evaluate in the next stage of engineering the use of backup air or consider the use of a motorized valve (with padlock)
28. Engineering should consider the distribution boxes of the flotation cells, as well as for the entire floatation area, nearby water points for cleaning.
29. The engineering shall review the structural design of the pulp distribution box to the flotation cells to confirm that they have considered the effect of vibrations and other forces.
30. The detailed engineering design should consider installing protective grating on top of the pump boxes.
31. The design should consider control options for the presence of foam inside the feed box.
32. For thickener rakes, as well as for the agitators of feed tanks to the filters, check the power supply back up system.
33. Engineering contemplates emergency sumps for the thickeners.
34. For areas of thickeners in the event of a spill, engineering includes containment such as gutters.

35. For the thickener filling pipes, the engineering considers the installation of a water line for cleaning.
36. For pumps in the event of an electrical failure, engineering contemplates the implementation of a failure position for the valves.
37. Prepare a maintenance plan for electrical failures.
38. When there is low level of suction pumps, consider level control, elaborate Standard Operating Procedure.
39. There is a back-up pump, but when there is a cooling failure you must review according to the maintenance plan.
40. When there is a low level or high level in the pump boxes, to avoid sedimentation or spillage, check the level control.
41. In all process areas, engineering has considered sump pumps for any spill or in the case any lines or equipment must be drained.
42. For rupture of filter plates, consider a sorting system in the pulp tank that feeds the filters.
43. Before a power outage, implement a procedure to restart the plant.
44. For faults in filter discharge belts, check stock of spare parts and corrective maintenance activities.
45. When a pump water seal fails, the design includes a backup pump. The envisioned water seal is intended to start and stop the pump so that the pump cannot run dry.
46. To prevent the seal water pump from operating due to inadequate selection, engineering should check pump selection.
47. Check attachment for the level sensor in the fresh water pumping system.
48. The need for the installation of by-pass lines on the control valves must be reviewed.
49. For the lime slurry system, the shut off temperature in the mill should be monitored. This will be reviewed with the seller of the lime plant, to consider installing a heat sensor to measure the temperature of the pulp in the discharge of the mill and install a thermocouple in the preparation tank.
50. The quality of the lime must be checked, the responsibility lies with the user and / or the supplier.
51. Minera Corani should be consulted for the technical specification of the flocculant.
52. Grating should be used in the flocculant preparation zone; engineering has considered the use of grating.
53. The accumulation of compressed air should be checked.

25 Interpretation and Conclusions

This section presents the main conclusions about the FEED stage developed by GMI. Highlights are presented as follow:

25.1 Property and Location

The land status of the Corani Project is a series of twelve (12) mineral claims or concessions. Mineral concessions in Peru are filed with the Instituto Nacional De Concessiones Y Catastro Minera (INACC) which is part of the Ministerio de Energía y Minas in Peru (MINEM). Claims can vary in size from 100 to 1,000 ha. They are rectangular geometries parallel to the UTM grid system employed in the district. The Corani Project is located in the districts of Corani, Macusani and Nuñoa, provinces of Carabaya and Melgar, department of Puno, in Peru, and covers an aggregate extent of 5,180.1213 hectares. The concessions are fully controlled by BCM and are free of any mortgage, lien, charge, royalty, or encumbrance. BCM controls the surface rights that cover the entire project area including the open pit, waste dump, process plant, water ponds, camp, and ancillary facilities required for operation. Surface rights total 2,424 hectares.

The twelve (12) mineral concessions comprising the Project are subject to compliance with payment of annual license fees in the amount of US$3.00 per hectare (“License Fees”). In addition, they are subject to an annual maintenance requirement with either of the following alternative obligations: minimum required levels of annual production of at least US$100 per hectare in gross sales (“Minimum Production”); or payment of an additional amount referred as Penalty of US$6.00 per hectare for the 7th through 11th year following the granting of the concession, and of US$20.00 per hectare thereafter; or exploration expenditures of 10 times the Penalty. Compliance with one of these three maintenance obligations, together with timely payment of License Fees, is required to them in good standing. Failure to comply with License Fee payments or Penalty payments for two consecutive years causes the forfeiture of the mineral concessions.

In the year 2018, the twelve (12) mineral concessions comprising the Project shall be subject to the obligations of Minimum Production, Penalties and exploration expenditures in accordance with the maintenance regime in force as of October 2008 whereby:

- The minimum production will be equivalent to one Tax Unit per year (approximately U.S. $1,333.00) per hectare granted for metallic minerals, and 10% of one Tax Unit per year per hectare granted for non-metallic minerals (“Minimum Production”);
- Failure to attain Minimum Production will trigger the obligation to pay a penalty equivalent to 10% of the Minimum Production per year per hectare, until the year in which the Minimum Production is attained;
- Year 2028 shall be the maximum deadline for the mineral concessions comprising the Corani Project to attain Minimum Production. Failure to do so will result in the forfeiture of these mineral concessions.

Control and current status were verified in August 2017 through an electronic database search of the Geologic Mining and Metallurgical Institute (INGEMMET). All concessions are in good standing.

25.2 Accessibility, Climate

The Project site is located in the eastern Andes mountain range, between 4,600 and 5,200 m above sea level (masl). The area is characterized by mountainous terrain dominated by volcanic rock above which sits glacial gravel. The lithologic and climatic conditions have given rise to a series of cirques or bowl-shaped, steep-walled basins. Apart from the vegetation associated with the wetlands mentioned above, areas of “puna” or alpine tussock grassland occupies the valleys and moderate to steep slopes. The areas above 4,700 masl mostly consist of steep mountainous slopes where erosion and climatic conditions largely prevent the development of soils or vegetation. These areas are scarcely vegetated with species specially evolved to withstand the harsh conditions. The naturally occurring acidic soils related to oxidation of sulfide bearing materials and the resulting ARD from exposed mineralized zones within the project area have also prevented the development of vegetation where these conditions occur.

Existing access to the Project site is primarily by road from the town of Macusani (located on the paved dual lane Interoceanic Highway), which is more readily accessible from the town of Juliaca, also serviced by commercial airlines from Lima. This route typically takes 4.5 to 5 hours. There are other access routes to the site from Cusco, taking approximately 6 hours by vehicle on increasingly primitive roads approaching the site. The City of Cusco is also serviced by commercial airlines from Lima.

The nearest town of significant size and with significant infrastructure is Macusani, which is the capital of the Province of Carabaya in the Region of Puno. Macusani is approximately 30 km east of the Project in a direct line. The access road from Huiquisa Bridge to the Permanent Camp will be improved. The length of the proposed Mine Access Road connecting the process Plant to Macusani is anticipated to be approximately 44 km. The Project site is located in the district of Corani, also in the Province of Carabaya. The closely orientated campesino communities of Chacaconiza and Quelcaya, which have a joint population of approximately 200 families (80 and 120 families, respectively), will be directly impacted by the mine development, in terms of landholding, rights to water, employment, etc. The community consultation undertaken with the Chacaconiza and Quelcaya communities to date has included a proposal for mining employment, generating widespread acceptance, mainly among younger community members, the teachers at local educational facilities, and community leaders. The current labor force is generally unskilled, mainly working on highway remediation and maintenance. A technical training program is planned to develop the skills of community members to fulfill employment requirements of the Project. The Project's requirement for labor will exceed the labor resources available in the Chacaconiza and Quelcaya communities. A ranking system will be developed with regard to geographical location of employment applicants, together with categorization and quantification of the labor force required.

25.3 History

Prior to the early 1950s, mineral exploration in the Corani district consisted of shallow prospect Pits and adits in the northern portion of the current project area. These prospects are of unknown age and may date back to colonial Spanish time. Antimony prospects south and east of the property reportedly were active in the early 1900s, when there was limited antimony production (C.R. Petersen, 1967).

The first modern evaluation of silver-lead mineralization began with the location of mineral concessions in 1951 and in 1956 Compañía Minera Korani was formed to develop the silver-lead mineralization previously prospected. The mines were developed and operated from 1956 to at least until 1967. Total historical production is uncertain, but is estimated at 100,000 t of silver-lead-zinc ore. In early 1967, estimated mine production was reported at about 3,400 short tons per month, with grades of 7.0-9.0% lead, 2.3% zinc, and 8.0 to 11.0 oz/ton silver (C.R. Petersen, 1967).

The next exploration activity was by a private Peruvian company, Minsur. That exploration was reported to include 40 shallow drill holes in various locations, including a number of close proximity holes in the gold zone (located south of the current resource area). Although Minsur is an active mining company in Peru; attempts by BCM to secure copies of Minsur's exploration data have been unsuccessful. None of Minsur's exploration information is available or verifiable; although reportedly gold mineralization was encountered in some of Minsur's drilling.

In late 2003 and early 2004, Rio Tinto Mining and Exploration began a surface exploration program for porphyry copper mineralization. That initial work by Rio Tinto defined anomalous silver and lead mineralization to the south of the Korani mines, and also defined a zone of anomalous gold mineralization in rock and soils. The concession ownership by Compañía Minera Korani apparently lapsed during the 1970s. The ownership of Minsur also lapsed prior to

Rio Tinto's exploration activities after 2000. Rio Tinto re-established some of the older concessions in their name beginning in 2003. BCM has added two concessions early in 2005 to create the current land position described in Section 4.

Six previous resource estimates and two previous mineral reserve estimated have been completed for the project and are published in previous technical reports beginning in 2006. Since 2006, the Measured and Indicated resource has grown from approximately 40 Moz of silver to over 300 Moz of silver.

25.4 Geological Setting and Mineralization

The Corani Project area is located in the northern part of Puno Department, southern Peru, within the Cordillera Oriental of the Central Andes. The Project area is underlain by Tertiary volcanic rocks of the Quenemari Formation, specifically a thick series of crystal-lithic tuffs and andesite flows, which overlie variably deformed Lower Paleozoic to Mesozoic metasediments of the Ambo and Tarma Groups. The primary host of mineralization is the Chacacuzina Member of the Quenemari Formation. The Chacacuzina is the youngest member of the Quenemari, and is comprised of a sequence of crystal-lithic and crystal-vitric-lithic tuffs. The tuffs are widely hydrothermally altered and pervasively argillized to low-temperature clays, and are variably faulted, fractured, and brecciated.

Mineralization at the Corani Project occurs in three distinct and separate zones: Corani Main, Corani Minas, and Corani Este, each differing slightly in character with regard to both alteration and mineral assemblages. In general, mineralization in outcrops throughout the Corani Project is associated with iron and manganese oxides, barite, and silica. Silicification is both pervasive and structurally controlled along veins. In drill core, the mineralization occurs in typical low to intermediate sulfidation Ag-Pb-Zn mineral assemblages. The most abundant silver-bearing mineral is fine-grained argentian tetrahedrite or freibergite.

Structurally, the Corani deposit is situated within a stacked sequence of listric normal faults striking dominantly north to north-northwest with moderate to shallow (50° to <10°) westerly dips. The hanging walls of the listric faults are extensively fractured and brecciated, providing the structural preparation for subsequent or syngenetic mineralization. The stacked listric faults are more prominent in the Corani Minas and Corani Main areas. The Corani Este area contains a single known listric fault with an extensively fractured and brecciated hanging wall. The contact with the underlying Paleozoic sediments corresponds locally to listric faults dipping shallowly to the west.

25.5 Deposit Types

The Corani deposit is best described as a low- to intermediate-sulfidation epithermal deposit with silver, lead, and zinc mineralization hosted in stock works, veins, and breccias. Mineralization is principally located in a set of lístric faults dipping west, with dilational segments related to subvertical structures and breccias in the hanging wall, and veinlets forming stockworks in the footwall. Structural control of the mineralization is a product of extensional tectonics that developed the series of north- to northwest-trending fractures and faults, and whose movements provided the structural preparation for the influx of mineralizing hydrothermal fluids.

Mineralization at Corani is likely both laterally and vertically distal to an intrusive fluid source. Mineral textures grade from coarse crystalline quartz-pyrite-chalcopyrite in the southern portion of the Project area, to finer grained, pyrite-dominated sulfide minerals in the north, suggesting a south-to-north hydrothermal fluid flow. This spatial zonation suggests a rapidly cooled ore fluid typical of a distal setting surrounding a buried intrusion. The multiphase nature of the mineralization and zonation at Corani may be related to multiple fluid exsolution events from an evolving porphyry type system that possibly underlies the southern part of the area. Alternatively, the mineralizing solutions may be related to shallow, subvolcanic dome emplacement.

25.6 Exploration

BCM began exploring the Corani Project in early 2005. In addition to drilling, exploration activities carried out by BCM include detailed geologic mapping, trenching, and geophysical surveying.

BCM has conducted general geologic surface mapping over the entire Project area. The total mapped surface is about 4.5 km wide (east-west) and 7.5 km long (north-south). In 2015, detailed surface mapping, including lithology, alteration, and structures, was performed at a scale of 1:2500 in the area of the proposed pits.

BCM has completed 25 trenches within the Project resource area (Corani Main, Minas, and Este) to verify the continuity of the structures covered by Quaternary sediments. Spacing between the trenches is roughly 50 to 100 meters. Channel samples from these trenches have produced an associated 1,295 assay intervals for a total of 2,924 meters of trench data.

VDG del Perú S.A.C. (VDG) conducted a ground geophysical campaign at the Corani Project on behalf of BCM in the fall of 2005. A total of 44.20 line-km of induced polarization (IP) data was collected, along with 50.95 line-km of magnetic survey. The geophysical surveys were aimed at assisting in geological mapping, including lithologies and key structures and at mapping mineralization and alteration associated with a low sulfidation gold-silver system. The objective of the IP/Res survey was to map the electrical response by means of high-resolution IP traverses across the favorable north-south corridor identified based on the results of both trench and drilling exploration. The field results of both methods were of good quality and were meaningful. The final chargeability and resistivity depth sections mapped systematically clear contrasts from line to line between the sub-surface and a nominal depth of 283 meters below surface. The chargeability outlined five (5) IP anomalies, two of which correspond to the Corani Main and Corani Este areas, respectively. Those anomalies accurately mapped the known mineralization and extended the size of both mineralized zones.

25.7 Drilling

Since 2005, BCM has completed a total of 556 drillholes at the Corani Project for a total of approximately 100,494.57 m. Drilling was completely by the Peruvian contractor, Bradly MDH primarily using LD250, JKS35, and LJ44 drill rigs. All of the drilling to date has been completed using diamond core drilling methods to produce either HQ (6.35 cm dia.) or NQ (4.76 cm dia.) core. Core recoveries are generally excellent, with no discernible variation in rate of recovery between the two core sizes (HQ and NQ). While on site, the QP carefully reviewed the drilling and sampling procedures employed by BCM with BCM staff. Based on that review, the QP finds no drilling, sampling, or recovery factors that might materially impact the accuracy or reliability of the drilling results.

Since 2006, BCM has undertaken to in-fill the known areas of mineralization in order to increase confidence in the resource estimate. This has been accomplished by in-fill drilling to produce a nominal 50-m drillhole spacing in previously more widely spaced drilling areas, with a focus on areas of higher grade mineralization. In general, drilling exploration has identified and further defined the distribution of mineralization within the three primary resource areas, Corani Main, Corani Este, and Corani Minas. Drilling results indicate that significant mineralization occurs in two basic forms; in large veins associated with the principal listric fault structures, and in stockwork veins found in the surrounding rocks.

25.8 Sample Preparation, Analyses and Security

BCM employs standard, basic procedures for both drill core and trench sample collection and analysis. Formal chain of custody procedures are maintained during all segments of sample transport. Samples prepared for transport to the laboratory are bagged and labelled in a manner which prevents tampering, and remain in BCM control until released to private transport carrier in Cusco or Juliaca. Upon receipt by the laboratory, samples are tracked by a blind sample number assigned and recorded by BCM. The samples are prepped according to ALS-Chemex preparation code PREP-31, and silver, lead, zinc, and copper assays are carried out by three-acid digestion followed by atomic absorption spectrophotometry (AA) analysis. Multi-element inductively coupled plasma (ICP) analysis is conducted on select sample intervals to assist with mineralization classifications and to guide the interpretation of the metallurgical process response.

BCM maintains an internal Quality Assurance/Quality Control (QA/QC) program which includes both standard and check (lab) sampling. GRE conducted a critical review of BCM's QA/QC program; toward that end, BCM provided GRE with QA/QC data in multiple Excel spreadsheet files. GRE compiled the data into a single, comprehensive QA/QC data worksheet for analysis and evaluation. Based on the results of GRE's review, in conjunction with observations and conversation with BCM personnel during the QP site visit, BCM's routine sample preparation, analytical procedures, and security measures are, in general, considered reasonable and adequate to ensure the validity and integrity of the data derived from BCM's sampling programs. GRE recommends that BCM expand the existing QA/QC program to include at least standards, blanks, and duplicates, and that QA/QC analysis be conducted on an on-going basis, including consistent acceptance/rejection tests. Each round of QA/QC analysis should be documented, and reports should include a discussion of the results and any corrective actions taken.

25.9 Data Verification

Data verification efforts included an on-site inspection of the Corani Project and core storage facility, check sampling, and manual and mechanical auditing of the Project database. Field observations during the site visit generally confirm previous reports on the geology of the Project area. Bedrock lithologies, alteration types, and significant structural features are all consistent with descriptions provided in existing Project reports, and the author did not see any evidence in the field that might significantly alter or refute the current interpretation of the local geologic setting.

Specific core intervals from 35 separate drill holes were selected for visual inspection and potential check sampling based on a preliminary review of the drill hole logs and associated assay values. In all cases, the degree of visible alteration and evidence of mineralization observed was generally consistent with the grade range indicated by the original assay value. Laboratory analysis was completed by ALS Peru using the same sample preparation and analytical procedures as were used for the original samples. Standard t-Test statistical analysis was completed to look for any significant difference between the original and check assay population means. The results of the t-Test showed no statistically significant difference between the means of the two trials (original versus check assay).

GRE completed a QAQC audit of the digital Project database by comparing a random selection of original assay certificates to the assay information contained in the Corani Project database. Results of the QAQC audit indicate a minor and acceptable error rate. GRE also completed a mechanical audit of the Project database in order to evaluate the integrity of data from a data entry perspective. Based on the results of the QP's check sampling effort, verification of drill hole collars in the field, visual examination of selected core intervals, and the results of both manual and mechanical database audit efforts, the QP considers the collar, lithology, and assay data contained in the Project data base to be reasonably accurate and suitable for use in estimating mineral resources and reserves.

25.10 Mineral Processing and Metallurgical Testing

Processing of the ore will be by conventional flotation recovery methods. The ore will be crushed close to the mine and the material conveyed to the processing plant, which will be approximately 500 m from the mine. The ore will be ground to 80% passing 106 microns in a SAG/Ball mill circuit. The material will then be floated with the rougher concentrates being reground to 80% passing 25 microns prior to cleaning to produce high-value separate lead-silver and zinc concentrates. Concentrates will be trucked to the port of Matarani for ocean shipment to smelters.

The following additional opportunities are identified with regard to mineral processing and material handling.

- Flotation optimization may improve metal recoveries. Testing under optimized conditions could increase recovery over that predicted by the geometallurgical model.
- Data generated during additional geotechnical drilling may show that it is feasible to steepen pit slopes.
- Higher concentrate grades may be achievable, which would increase the net smelter returns.
- Operating cost improvements may be derived from using conveyor systems to transport tailings to the disposal sites.

25.11 Mineral Resource Estimates

GRE estimated the Mineral Resources for the Corani project during the first quarter of 2015. No new drilling, geology, or metallurgical test work has been performed since then. That work and the resulting Mineral Resources were documented and published in the May 30, 2015 Technical Report. The 2015 mineral resource block model was used for estimation of the Mineral Resources and Mineral Reserves of the Corani Project in the current Corani Project Detailed Engineering Phase 1 (FEED) Technical Report.

The resource model has three main lithologies: basement sediment with minor quantities of mineralization, the mineralized (pre-mineral) tuff, and a mostly unmineralized post mineral tuff which is assumed to be barren. Mineralization has been defined by 7 mineralization types, which were later grouped into, oxidized, transition, and sulphide groups. The Mineral Resources for the Corani project are shown inTable 25-1. The Mineral Resources were generated within the $30.00/troy ounce silver, $1.425/lb lead, and $1.50/lb zinc price Whittle pit shell and the calculated $11/tonne NSR cut-off. Table 25-2 shows the potentially leachable Mineral Resource contained within the Whittle pit shell at a 15 g/t cut-off that is available in addition to the Mineral Resource shown in Table 25-1.

Table 25-1: Total Mineral Resources (Includes Both Resources and Reserves)

Category	Ktonnes	Silver gpt	Lead %	Zinc %	Silver Million oz	Lead Million lb	Zinc Million lb
Measured	29,209	56.2	0.912	0.582	52.8	587	375
Indicated	181,902	40.7	0.741	0.495	238	2971.3	1983.5
Measured + Indicated	211,111	42.8	0.765	0.507	291	3,558	2,359
Inferred	31,231	40.6	0.742	0.512	40.8	510.6	352.4

Note: Cut-off Value : $11.00/tonne covers process and general and administrative costs.

Table 25-2: Total Mineral Resource of Potentially Leachable Material

Category	Ktonnes	Silver gpt	Silver Million oz
Measured	5,006	38.0	6.12
Indicated	19,690	23.1	14.61
Measured + Indicated	24,697	26.1	20.72
Inferred	8,722	25.1	7.03

25.12 Mineral Reserve Estimates

The Mineral Reserve Estimate is based on the 2015 GRE resource block model, using updated Whittle optimization parameters, new pit designs, and new phase designs.

The project Mineral Reserves consider only measured and indicated resource categories, which have been converted to proven and probable reserves categories, respectively. Mineral Reserves are defined as being the material to be fed to the process plant in the mine plan already described, and are demonstrated to be economically viable in the Detailed Design Phase 1 (FEED) economic model.

Table 25-3: Corani Project Mineral Reserves

Classification	Tonnes Mt (dry)	Grade			NSR $/t	Contained Metal		
		Silver g/t	Lead %	Zinc %		Silver M oz	Lead M lb	Zinc M lb
Proven	20.8	65.8	1.03	0.71	37.17	44	472	323
Probable	118.3	47.5	0.87	0.57	28.55	181	2,274	1,486
Total Proven + Probable	139.1	50.3	0.90	0.59	29.84	225	2,746	1,809

Notes:

8) The Mineral Reserves have been estimated using the definitions of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

9) The Mineral Reserves have been estimated using the following metal prices: $20.00/oz Ag, $1.00/lb Zn, $0.95/lb Pb using a revenue factor 1.00 pit shell as a basis for the pit design.

10) Only pre-mineral tuff type of material has been considered as reserves.

11) NSR Cut-off grades used are equal or higher than: $11.11/t for the East Pit, and $11.26/t for Minas and Main pits.

12) The effective date for these Mineral Reserves is 1 May 2017.

13) Totals / Averages may not add up due to rounding of individual tonnes and grades.

14) The tonnes and grades shown above are considered a Mineral Reserve because they have been demonstrated to be economically viable through the FEED study financial model using the following metal prices: $18.00/oz Ag, $1.10/lb Zn, $0.95/lb Pb.

Additional geotechnical drilling should be completed within the planned pit. This will confirm the current pit slope design basis, and potentially allow an increase in the pit slope angles. After drilling, testing and analysis are required to design the final pit slopes. Boreholes should be completed as monitoring wells, and multiple-well aquifer testing will be performed to better assess the dewatering requirements for the material. The analysis for the pit slope design should consider a fault geological-structural model in 3D using the database from the geological drilling.

25.13 Mining Methods

The Corani Project will be mined using conventional open pit mining methods, with either an owner mine equipment or a contractor mining scenario. The base case assumes contractor mining. The rock will be broken by drilling 0.156-m diameter blast holes and blasting with ANFO and emulsion. Broken rock will then be loaded into 130-136 tonne trucks using a 13 m^3 front end loader or one of two 15 m^3 hydraulic shovels. Support equipment includes two D10 bulldozers, a road grader, water trucks, rubber tire dozer, compactor, excavator, fuel and lube trucks, and other miscellaneous equipment.

During a 10-month pre-production stripping, pioneering, haul road construction phase prior to plant construction, 4.7 million tonnes of waste rock will be mined to generate construction material. Another 7.6 million tonnes will be mined immediately prior to production. The mine is designed to generate 7.875 million tonnes of ore per year with strip ratios of 3:1 to 4:1 during the first two years, falling to below 2:1 the third year, and then below 1:1 the final 3 years of the mine life.

25.14 Recovery Methods

The process plant has been designed to treat 22,500 t/d of ore, operating 350 days per year and an average availability of 92%.

For this purpose, the following process stages have been designed:

- Primary Crushing and Coarse Ore Stockpile
- SAG – Ball Mill
- Sequential Lead and Zinc Flotation
- Concentrate Thickening and Filtering (Pb, Zn)
- Tailings Thickening and Filtering Plant
- Dry Tailings Disposal Plant
- Reagents and Utilities Areas.

The plant layout has been optimised in area and costs.

The design considers dust suppression systems in the crushing and coarse ore stockpile. The thickening and filtration for both, concentrates and tailings, have considered sump areas for containing and rehandling of the spills.

The engineering also has incorporated the filtration of tailings and the transport to the waste deposit area.

The design was based on the mining plan available at the beginning of the FEED, therefore it is required to up-date the engineering using the current mining schedule issued by BCM.

In conclusion the process plant has a flexible design that allows to work in different operational conditions, noting the following:

- By-pass for concentrates from Rougher and 1st Cleaner Lead flotation
- By-pass for Zinc flotation circuit to the final tailings thickener, between others.

The design also has in-line analyzers systems in both flotation circuits (lead and zinc) allowing to operate with higher sensibility with regards to the feeding grades to the plant.

The specific power consumption for the process plant is 42.3 kWh/t of treated ore.

Finally, it is concluded that there is a need to work with an intelligent treatment system, the distribution of recovered water, how to optimize its use and minimize negative impacts on the recovery of silver in the flotation stages.

25.15 Environmental Studies, Permitting and Social or Community Impact

Additional laboratory testing on samples of waste rock, filtered tailings and the mix of them, should be performed, such as shear strength in drained and undrained conditions, characteristic curve for unsaturated permeability, resonant column and torsional shear for dynamic properties. The laboratory testing results will allow a better understanding of the behavior of those materials, alone and mixed, such as drained and undrained conditions, unsaturated seepage by rainfall, seismic response and so on, for a potential optimization of the Main Dump and pit backfill design.

A sediment control plan should be implemented for avoiding the premature reduction of the Plant Water Pond capacity. The site wide water balance should be continuously updated based on an updated hydrology and plant parameters, such as natural and residual moisture of the waste rock and tailings, mine plan, production rate, production increase, water requirement, among others. In addition, monitoring station of solid and liquid flows should be implemented downstream the Plant Water Pond.

Because of the unfavorable geotechnical characteristics of the Plant Water Pond foundation, a good quality control and quality assurance program should be implemented during construction, including geoelectrical surveying for verifying potential geomembrane defects which produce leaks.

A monitoring plan, monitoring instrument alert levels and operation and contingency manual, should be developed for the whole mine facilities. A best industry practice should be applied in the development of those documents, mainly for the critical components. Those documents and manuals will support the routine operation and will produce a safe environment.

A group of specialists, including geotechnical, hydrologist and mechanical engineers, should carried out Annual Safety Reviews of the whole facilities, taking into consideration the Canadian Dam Association guidelines. The Safety Report should update the monitoring plan, operation and contingency manual, stability condition, water management, critical equipment operation of each mine facility and provide action lists for improving the functioning of them. Regardless the Annual Safety Review, all the facilities will have to be inspected after the occurrence of an extreme storm or earthquake event.

25.16 Market Studies and Contract

Penfold Limited conducted a review of the lead and zinc concentrate markets, smelting charges, penalties, concentrate handling, and land and ocean transportation costs. The supplied information was used as a guide to develop all associated payments and expenses associated with the sale of Corani concentrates. There are no letters of intent or sale agreements in place. All information is based on Penfold's experience for similar concentrates.

25.17 Economic Analysis

The economic analysis was performed using a Discounted Cash Flow (DCF) which is a standard industry practice. The key assumptions used for the study are shown in Table 25-4 and establish a "Base Case". The table provided the life-of-project averages for grade recovery and these values vary over the life of the project depending on the head grades and split between mixed sulfide ore and transition ore.

Table 25-4: Key Assumptions for the Corani Project – Base Case

Annual ore production – years 1 to end of life (ktonnes)	7,875
Overall process recovery – silver – into both lead and zinc cons	69.9%
Overall process recovery – lead – into lead cons	61.1%
Overall process recovery – zinc – into zinc cons	67.1%
Total processed ktonnes	139,073
Average silver grade (g/t)	50.3 g/t
Average lead grade (%)	0.90%
Average zinc grade (%)	0.59%
Payable ounces of silver net of smelter payment terms (total)	144 million
Payable pounds of lead net of smelter payment terms (total)	1.58 billion
Payable pounds of zinc net of smelter payment terms (total)	1.03 billion
Overall stripping ratio	1.49 to 1
Life-of-Mine years	18

The results of the economic analysis for the project indicate an after-tax internal rate of return (IRR) of 15.1% and net present value (NPV) of $404.5 million at a 5% discount rate based on metal prices of $18.00 per ounce silver, $0.95 per pound for lead, and $1.10 per pound zinc.

25.18 Adjacent Properties

There are no adjacent mineral properties which might materially affect the interpretation or evaluation of the mineralization or exploration targets of the Corani Project.

25.19 Other Relevant Data and Information

Bear Creek and GMI have created a Project Execution Plan and have conducted a Hazard and Operability study, creating a project development pathway designed to minimize risk and uncertainty, manage construction performance and schedule, to deliver the project on budget. The project is planned to be constructed over a three-year time span with engineering continuing through 2017, access road, plant equipment procurement and fabrication beginning the first quarter of 2018. Commissioning and start up is scheduled for the first quarter of 2021.

25.20 Capital and Operating Costs

The Capital cost estimate (CAPEX) presented in this Report is for a silver-lead-zinc mine and concentrator capable of producing and processing an average of 22,500 tpd of ore (dry basis). The total life of mine capital investment for the Corani Project is estimated to be $585.6 million.

The initial CAPEX includes the design, permitting, pre-stripping, construction, and commissioning of the mine, plant facilities, ancillary facilities, utilities and camp. The CAPEX also includes costs for engineering, construction management, and Owner's costs.

The total life of mine operating cost is estimated at $3,295.6 million. Estimated life of mine average plant operating and maintenance cost is 10.11 USD/t ROM.

26 Recommendations

Recommendations within this section relate to specific portions of this document's evaluations and studies. Identified items present opportunities to improve the understanding of technical issues associated with the project as well as reduce risk. Multiple identified options present opportunities to improve the project. The detailed scope of work will be developed as part of the front-end Detailed Engineering effort. A preliminary estimate of costs associated with this effort is included at the end of this section.

26.1 Geological Setting and Mineralization

26.1.1 Geotechnical

Monitoring, Operation, and Contingency Plans

A monitoring plan, monitoring instrument alert levels and operation and contingency manual, should be developed for the whole mine facilities. A best industry practice should be applied in the development of those documents, mainly for the critical components. Those documents and manuals will support the routine operation and will produce a safe environment.

Plant Water Pond

Because of the geotechnical characteristics of the Plant Water Pond foundation, a good quality control and quality assurance program should be implemented during construction, including geoelectrical surveying for verifying potential geomembrane defects which produce leaks.

A group of specialists, including geotechnical, hydrologist and mechanical engineers, should carry out Annual Safety Reviews, taking into consideration the Canadian Dam Association guidelines. The Safety Report should update the monitoring plan, operation and contingency manual, stability condition, water management, critical equipment operation and provide action lists for improving the functioning of them. Regardless the Annual Safety Review, all the facilities will have to be inspected after the occurrence of an extreme storm or earthquake event.

Main Dump

The Main Dump footprint expands the previously proposed waste dump footprint. Construction and reclamation materials are available within the dump footprint. Characterization of these materials, and assessment of their ability to satisfy project requirements should be carried out as part of the early detailed design work. Foundation strength characterization (including undisturbed sampling and strength testing), detailed design of the seepage collection pond and pumping system, and additional construction material characterization are required. Limited tailings filtering and geotechnical tests were completed as part of the Feasibility Study.

Additional testing should be completed to confirm the allowable tailings moisture content, the ability to filter the tailings, and final drain-down and stability characteristics (including testing of filter tailings dynamic/seismic properties). Updated geotechnical modeling and a detailed placement sequence for co-disposal and PAG encapsulation must be performed based on new field and laboratory data. Finally, the consolidation of tailings under loading must be ascertained to determine if excess pore pressure conditions are occurring and to determine the seepage behaviour of tailings under geostatic loading. These analyses must be performed for the Main Dump and for the Este pit backfill area.

In the next design stage characterization tests on filtered tailings and tailings mixed with mine waste simulating conditions which will occur in the stockpile should be carried out.

- It is recommended that dynamic resonant column and torsional cut tests be carried out on a mixture (1 to 1) of the mine waste and filtered tailings to determine the actual behavior of the materials in the stockpile. This test also allows estimation in laboratory of the velocity profile of the material.

- It is also recommended that triaxial tests of a 1: 1 mixture of filtered tailings and mine waste to assess the characteristics of the co-disposition of the two materials.

- It is recommended that type MASW and MAM to geophysical tests be completed obtain the profile of velocities in the field, once the deposit begins operations. Although conservative parameters have been used in the 1D response analysis, this will determine any behavior not observed in the existing documentation, which may generate potential problems during the operation.

- Consideration should be given to the recommendations presented and incorporated in the seismic design of pile as the design represents a deposit of up to 300 m height measured from the bottom.

- The 2D physical stability evaluation using the equilibrium method analyzed two critical sections of the deposit under static and pseudo-static conditions, the latter for a return time of 475 years. The safety factors resulting from this analysis were higher than the minimums recommended in the design criteria. Also, the permanent displacements induced by earthquakes were smaller than the allowable. Both criteria verify the stability of the design presented. The parameters used in the analysis must be validated with above laboratory tests.

- Installation of geotechnical instrumentation consisting of: thirty-eight survey monuments, two robotic stations, two vertical inclinometers, four vibration string piezometers, two open tube piezometers and one accelerometer is recommended. The objective of the instrumentation is to monitor displacement, groundwater level and water quality control, to verify the behavior of the pile and estimate any potential problems that may arise during the operation. The survey monuments will be located on the slopes and on the summit crest.

During the construction stage it is recommended that the properties of the unsuitable materials be monitored.

- A percolation analysis using a two-dimensional model of finite elements of constant or variable hydraulic load, will allow evaluation of permeabilities in saturated and unsaturated conditions for the materials that compose the section.

- Estimation of the undrained shear strength of the material by one of the following alternatives should be carried out: 1) Laboratory Test - Unconfirmed Triaxial No Drain (UU); 2) Field Test – CPTu; 3) Field Test - Torvane which recommends the installation of two piezometers in order to constantly monitor the piezometric surface and the flow conditions in the deposit. One of the piezometers will be installed in the deposit and the other in the containment dam.

- The surface water management plan should include a diversion of runoff around the outside the pile during material placement and operation. Careful construction, placing more highly saturated material in the back of the pile and less saturated material in the front is also recommended.

Topsoil

During the construction stage it is recommended that the properties of the topsoil material be monitored.

- Estimation of the undrained shear strength of the material by one of the following alternatives should be carried out: 1) Laboratory Test - Unconfirmed Triaxial No Drain (UU); 2) Field Test – CPTu; 3) Field Test - Torvane which recommends the installation of two piezometers in order to constantly monitor the piezometric surface and the flow conditions in the deposit. One of the piezometers will be installed in the deposit and the other in the containment dam.

- It is also recommended that two piezometers be installed to constantly monitor the piezometric surface and the flow conditions in the Topsoil Dump. One of the piezometers should be installed in the deposit, and the other on the containment dam.
- It is also recommended that survey monuments be installed to monitor possible settlements and / or displacements in the pile. These survey monuments will be located on the slopes and on the summit crest.
- The surface water management plan should include a diversion of runoff around the outside the pile during material placement and operation. Careful construction, placing more highly saturated material in the back of the pile and less saturated material in the front is also recommended.

Mine

Additional geotechnical drilling should be completed within the planned pit. This will confirm the current pit slope design basis and potentially allow an increase in the pit slope angles. The pit will intersect the unconsolidated sediments lining the floor of the upper bofedal and lower bofedal areas. Additional drilling, testing, and analyses are required to design the pit slopes within the bofedal soils and to develop a detailed plan for dewatering and mining the bofedal soil material. This will involve drilling several boreholes through the unconsolidated sediments, with production of detailed stratigraphic logs and undisturbed sampling for density and strength testing of the unconsolidated material. Boreholes would be completed as monitoring wells, and multiple-well aquifer testing will be performed to better assess the dewatering requirements for the material. Detailed pit slope design and soil mining plans must then be developed.

26.2 Exploration

GRE recommends that BCM produce annual (or seasonal) exploration reports to describe the drilling and sampling carried out during each given year or drilling campaign. The exploration report should contain adequate detail concerning the drill rig, drilling contractor, number of holes, total meters, recovery rates, drill targets, and rationale for drill hole distribution, etc., to ensure that all pertinent information is captured and catalogued in a practical and efficient manner for ease of future use.

26.3 Data Verification, Sample Preparation, Analysis, & Security

GRE recommends that BCM establish a routine, internal mechanical audit procedure to check for overlaps, gaps, total drill hole length inconsistencies, non-numeric assay values, and negative numbers. The internal mechanical audit should be carried out after any significant update to the database, and the results of each audit, including any corrective actions taken, should be documented and stored for future use in database validation.

Based on the positive assay results of the selected intervals of previously un-sampled post-mineral tuff, the QP recommends that BCM sample at least one 2-m interval for every 20 m drilled and logged as post-mineral tuff. If positive assay results are returned, additional intervals should be selected accordingly to ensure that all mineralized material is analyzed.

Based on observations and conversation with BCM personnel during the QP site visit, in conjunction with the results of GRE's review and evaluation of BCM's QA/QC program, the QP makes the following recommendations:

- Formal, written procedures for data collection and handling should be developed and made available to BCM field personnel. These should include procedures and protocols for field work, geological mapping and logging, database construction, sample chain of custody, and documentation trail. These procedures should also include detailed and specific QA/QC procedures for analytical work, including acceptance/rejection criteria for batches of samples.

- A detailed review of field practices and sample collection procedures should be performed on a regular basis to ensure that the correct procedures and protocols are being followed.

- Review and evaluation of laboratory work should be an on-going process, including occasional visits to the laboratories involved.

- BCM's existing QA/QC program should be expanded to include at least standards, blanks, and duplicates. All QA/QC control samples sent for analysis should be blind, meaning that the laboratory should not be able to differentiate a check sample from the regular sample stream. The minimum control unit for check sample insertion rate should be the batch of samples originally sent to the laboratory. Samples should be controlled on a batch by batch basis and rejection criteria should be enforced. Ideally, assuming a 40-sample batch, the following control samples should be sent to the primary laboratory:

  - Two blanks (5% of the total number of samples). Of these, one coarse blank should be inserted for every 4^{th} blank inserted (25% of the total number of blanks inserted).

  - Two pulp duplicates (5% of the total number of samples).

  - Two coarse duplicates (5% of the total number of samples).

  - Two standards appropriate to the expected grade of the batch of samples (5% of the total number of samples).

- For drill hole samples, the control samples sent to a second (check) laboratory should be from pulp duplicates in all cases and should include one blank, two sample pulps, and one standard for every 40-sample batch.

- The purpose of the coarse duplicates is to quantify the variances introduced into the assay grade by errors at different sample preparation stages. Coarse duplicates are inserted into the primary sample stream to provide an estimate of the sum of the assay variance plus the sample preparation variance up to the primary crushing stage. An alternative to the coarse duplicate is the field duplicate, which, in the case of core samples, is a duplicate from the core box (i.e., a quarter core or the other half core). Because sample preparation is currently carried out by the laboratory (and not by BCM), if coarse duplicates are preferred (to preserve drill core), the coarse duplicates should be sent for preparation and assaying by the second laboratory.

- QA/QC analysis should be conducted on an on-going basis and should include consistent acceptance/rejection tests. Each round of QA/QC analysis should be documented, and reports should include a discussion of the results and any corrective actions taken.

26.4 Mineral Processing and Metallurgical Testing

Additional metallurgical testing should be completed to optimize the known flotation test conditions. The reagent scheme should be optimized, and additional testing could explore the possibility of higher $ZnSO_4$ dosages on difficult samples in addition to removing and/or reducing other depressants.

The geometallurgical model was developed using all metallurgical testing data, including tests representing non-optimized conditions. Once additional testing has been performed, and samples representing optimized test conditions are available, the statistical model should be re-evaluated to ensure estimated recoveries represent optimal conditions.

The selected process flowsheet should be re-tested to confirm:

- Primary grind size
- Regrind size
- Residence time for each flotation stage
- Reagent selection and dosage

Additional lock cycle testing is recommended for each deposit, particularly material representing moderate to low zinc grades, which is under-represented in the current test database. This will allow for validation of the final estimated recoveries and the selected concentrate grades. This testing should include analysis of minor elements; limited test data is available regarding the concentration of minor elements in the final concentrate.

The geotechnical testwork on filtered tailings mentioned in Section 26.1 may lead to the need for additional tailing filtration testing. The filtration equipment needed for mechanical placement of tailings is a significant capital and operating cost to the project. It will be important to determine the filtered tailing physical properties so that the equipment can be specified to achieve the required results.

Testing of an acid brine leaching process for oxide mineral resources should be considered, or testing of an alternative flowsheet, for the non-floatable resource areas of the deposit. If this were successful, it could potentially add value to the project.

26.5 Mineral Resource Estimates

The 2011 Feasibility Study recommended review of the check assay and standards because significant scatter was observed in that data. This issue has not been fully resolved, and future drill programs should work to address the issue.

26.6 Mining Methods

GRE recommends BCM obtain formal bids from several mine contractors and mine equipment quotations from equipment manufactures' including MARC quotes in order to more fully evaluate the owner mining and contractor mining alternatives.

26.7 Recovery Methods

- The design of the plant must be updated considering the latest mining plan

- Confirm the burden and spacing of blasting to reduce the stationary grizzly oversize (1 m opening) and the rock breaker duty.

- Optimize the feed and discharge primary crusher bin capacities to match with the haul truck size.

- Evaluate the need to include a cover for the coarse ore stockpile based on environmental considerations.

- Complete further detailed design on the pebble handling system once more data is obtained on their effects on SAG mill operation.

- Incorporate metallurgical samplers in lead and zinc flotation feed circuits and both final concentrates and tailings to be able to realize auditable mass balances for silver, lead, and zinc.

- Carry out new tests using SFR flotation cells with the purpose of reducing the amount of penalty elements (As, Sb, clays) and maximising the silver recovery in the lead concentrate.

- Incorporate two independent process water circuits for lead and zinc flotations to remove the potential detrimental effects recirculated water may have the lead flotation efficiency and further ensure optimum lead and silver recovery. This could potentially reduce the reagent consumption and result in operating cost savings for the process plant.

- Incorporate a hydrogen peroxide and activated carbon in-line additions into the process plant design.

- Incorporate a sulphuric acid addition to better control the pH of the recirculated water.

- Carry out a dynamic simulation to detect possible bottle necks in the plant. This should be done as one of the first steps of the continued detail engineering design.

26.8 Power Supply

The project electrical supply should be designed to be independent of the other possible community electrification requirements related to the Project in order to ensure operational reliability.

26.8.1 Risks

The following risks have been identified:

- The high altitude of the site may have a greater-than-expected negative impact on worker productivity.
- The high altitude of the site may result in greater-than-expected impacts on the function and capacity of diesel-powered equipment and electrical components.
- As with any large-scale mine development, there is a risk that additional capital may be difficult to raise in the event that costs increase during the pre-production period.
- A currency exchange risk exists. A weakening of the Peruvian Nuevo Sol (PEN) would lower the cost of in-country expenses. However imported materials, priced in USD, would become higher. During operations, the sale of metals, priced in USD, would benefit from a weaker PEN. A strengthening PEN would have the opposite effects. It is also important to note that BCM holds cash and short-term investments priced in USD that will help offset risk associated with the exchange rate.
- Although local communities have generally supported the Project development, there is a risk that sentiments could change, or that special interest groups from outside the community could mobilize opposition to the Project.
- During operations, a potential silver migration from the lead flotation circuit to zinc flotation circuit must be evaluated with additional metallurgical testing.

26.9 Market Studies and Contract

The smelters selected for the study should be contacted to verify their capacity and ability to accept the proposed quantity and quality of produced lead and zinc concentrates. As part of the program, additional concentrate analysis should be completed to further define the concentrate qualities. Letters of intent would be desirable for future project financing and associated due diligence along with firm quotes from the smelters. In addition, firm quotes should be obtained to confirm the transportation costs for the concentrate to the Matarani port as well as port handling costs. Under current market conditions, there may be an opportunity to improve business terms beyond the assumptions used in this study.

26.10 Environmental Studies, Permitting and Social or Community Impact

In 2016, the Ministry of Energy and Mines approved the modification of the ESIA based on the Optimized and Final Feasibility Study prepared by 2015. Another modification of the ESIA based on the FEED study is required and should be pursued by Bear Creek in the short term.

In addition, it is recommended that Bear Creek commence the permitting process on water rights and mine plan approval, both of which are critical, early-stage permits. Bear Creek is also encouraged to utilize all efforts in maintaining its social license and ensuring the continued strong support from local communities, local and regional governments, and the central Peruvian government.

26.11 Capital and Operating Cost

26.11.1 Optimization

It may be possible to reduce the cost of delivering tailings to the Main Dump and pit backfill disposal sites by varying the proportion of tailings delivered by conveyor systems and by trucks during the period when tailings are being produced. It is recommended that an optimization study be carried out to determine this, and a detailed plan should be devised. Tailings will be co-disposed with waste rock in the Main Dump and pit backfills. In general, it is expected that it will be cheaper to use conveyors instead of trucks to deliver tailings to ultimate disposal destinations, but exclusive use of conveyors may be less practical for tailings destined for the pit backfill. During the period when pit backfilling will be taking place, the current schedule indicates that truck capacity will be available; therefore, an optimization study should specify the ideal mix of conveyor/truck transport of tailings over time, depending partly on truck availability.

A study should be conducted to match operating equipment to the high-altitude conditions, potentially identifying equipment outfitted with pressurized cabs and other worker comfort and performance additions. Caterpillar equipment offers high-altitude arrangements (HAA), and these modifications allow their power ratings to be valid to 4,877 m.

26.12 Other Relevant Data and Information

Complete the Project Execution Plan (PEP) and project risk assessment called a Hazard and Operability study (HAZOP) as described in Section 24.

26.13 Constructability

Generation of a site-wide cut-and-fill material balance is recommended, including specialized construction materials such as clays, concrete aggregate, drain rock, road base, and rip rap.

26.14 Water Balance

The probabilistic water balance employs models from 2015 to predict pit dewatering and main dump seepage quantities. These models are expected to be sufficiently accurate for this level of study; however, they must be updated to reflect the revised mine plan and the planned modifications of the main dump construction methods. Once these models are revised and new results are obtained, the water balance must be remodelled to ensure the project can meet the required water quality and water quantity commitments in the ESIA. The surface water management plan will likely require updates to meet the new water balance.

26.15 Estimated Cost for Recommended Work

During the course of the work on this Project, the contributors have developed several recommendations for future consideration and execution by BCM. The estimated costs to complete the recommended work are shown in Table 25-5. The total of these costs represents the anticipated cost of the FEED study through its completion.

Table 25-5: Estimated Costs for Recommended Work

Item	Estimated Cost
Main Dump & Topsoil Geotechnical	$100,000
Mine Geotechnical	$1,100,000
Metallurgy	$1,310,000
Constructability	$160,000
Water Balance	$200,000
Environmental / Social / Permitting	$300,000
Completion Detailed Engineering including Optimizations	$6,830,000
Total	**$10,000,000**

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